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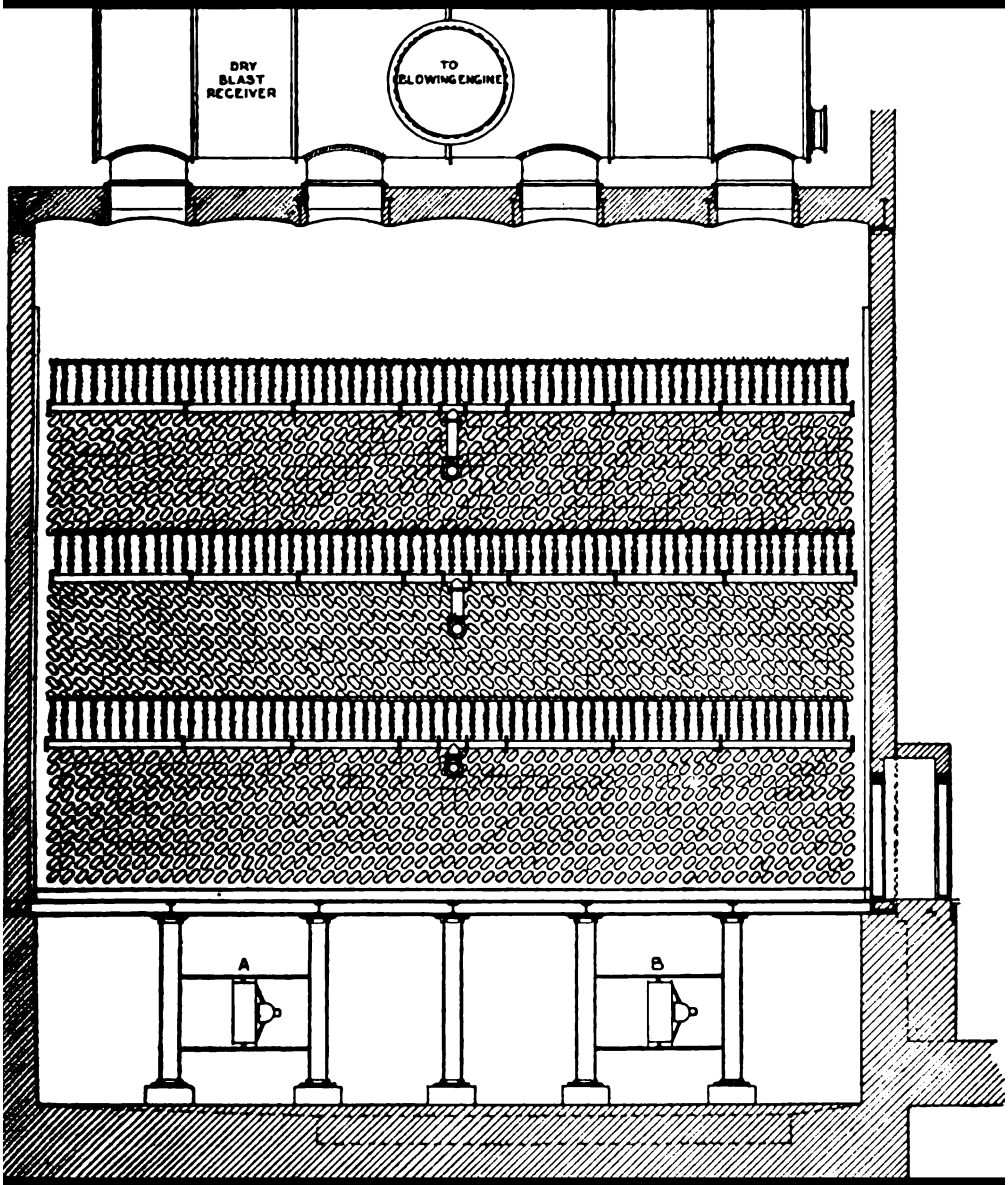

Mining and metallurgy

American Institute of Mining,
Metallurgical, and Petroleum Engineers

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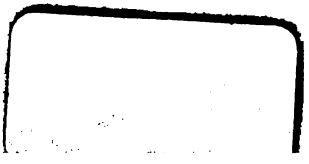

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Bi-Monthly Bulletin

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

ORGANIZED 1871

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JANUARY

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SECTION I.

INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions.*

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American Institute of Mining Engineers.

OFFICERS.

For the year ending December 29, 1904.

[This list was modified December 29, 1904, by sundry resignations, offered and accepted, and by sundry appointments to fill vacancies thus created, all of which was done to facilitate the legal incorporation of the Institute. A detailed statement of these changes and of the subsequent proceedings will be officially communicated to members hereafter; it being impracticable to delay the issue of the present number of this Bulletin for the preparation of such a statement.]

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Vice-Presidents.

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T. A. RICKARD,*.....New York, N. Y.
JAMES F. KEMP,.....New York, N. Y.

(Term expires February, 1905.)

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C. D. WALCOTT.....Washington, D. C.
GEORGE W. MAYNARD.....New York, N. Y.

(Term expires February, 1906.)

Managers.

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JAMES W. NEILL.....Salt Lake City, Utah.
M. D. VALENTINE.....Woodbridge, N. J.

(Term expires February, 1905.)

HEINRICH RIES.....Ithaca, N. Y.
B. B. LAWRENCE.....New York, N. Y.
F. KLEPETKO.....New York, N. Y.

(Term expires February, 1906.)

F. L. GRAMMER.....Baltimore, Md.
CHARLES H. SNOW.....New York, N. Y.
JOSEPH HARTSHORNE.....Pottstown, Pa.

(Term expires February, 1907.)

Secretary.

R. W. RAYMOND,
99 John St., New York, N. Y.

Treasurer.

FRANK LYMAN,
New York, N. Y.

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* By appointment of the Council, to fill the vacancy caused by the death of Mr. Philip W. Moen, September 12, 1904.

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Bi-Monthly Bulletin.

Beginning with the present number of the Bi-Monthly Bulletin (No. 1, January, 1905), the papers which have heretofore been distributed separately in pamphlet-form to members will be issued in the form of a bi-monthly publication. This change has been made for the purpose of securing the advantage in postal rates given by the U. S. Post Office to periodicals. Since, for many years past, the pamphlet-batches have been sent to members in alternate months, the new arrangement will involve no practical difference to them.

A small stock of separate papers will be reserved for those who desire extra copies of any single paper.

All communications concerning the contents of this Bulletin should be addressed to R. W. Raymond, Secretary, 99 John St., New York City (P. O. Box 223; Telephone number 5477 John).

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Library of the American Institute of Mining Engineers.

The library now comprises approximately 14,000 volumes, 4,000 pamphlets and 650 maps, which are available at the library rooms, 99 John Street, New York City, to members of the Institute and others properly introduced.

During the past year a catalogue of the periodicals and other publications in the library of the American Institute of Mining Engineers, together with a similar catalogue of the publications in the library of the American Institute of Electrical Engineers, was sent to all members of both societies, in the hope that sufficient interest would be created to secure the volumes now lacking. Until further notice, a list of all additions to the library, and suitable acknowledgment of donations of books, will be published in each issue of the Bi-Monthly Bulletin.

Accessions.

October 4 to December 15, 1904.

DONORS AND DONATIONS.

FROM AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
Transactions, vol. xxi., 8°, New York, 1904.

" AMERICAN INSTITUTE OF MINING ENGINEERS.
Transactions, vol. xxxiv., 8°, New York, 1904.

" AMERICAN IRON AND STEEL ASSOCIATION.
Annual Statistical Report, 1903, 8°, Philadelphia, 1904.

" ATKINSON, TOPEKA & SANTA FÉ RAILROAD.
Annual Report, 9th, 8°, 1904.

" BELGIUM—MINISTÈRE DE L'INTERIEUR ET DE L'INSTRUCTION PUBLIQUE.
Annuaire statistique. Years 28, 30–31 and 34, 8°, Bruxelles, 1898, 1900–'01, '04.
Catalogue de la Bibliothèque de la Commission centrale le statistique, vol. ii., 8°, Bruxelles, 1904.

" BRITISH MUSEUM LIBRARY.
Catalogue of Early English Books to 1640, 3 vols., 8°, London, 1884.

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FROM BRITISH MUSEUM LIBRARY.

Subject Index of Modern Works Added in 1881–1900. Edited by G. K. Fortescue. 3 vols., 8°, London, 1902–'03.

" BRITISH COLUMBIA—BUREAU OF PROVINCIAL INFORMATION.
Mining in British Columbia, 8°, Victoria, 1904.

" CANADA, DEPARTMENT OF INTERIOR.
Report of the Commission Appointed to Investigate the Different Electro-Thermic Processes for the Smelting of Iron-Ores and the Making of Steel in Operation in Europe, 8°, Ottawa, 1904. (2 copies.)

" CANADA GEOLOGICAL SURVEY.
Annual Report, vol. xiii., 8°, Ottawa, 1903.

" R. C. CHISM.
Encyclopædia of Mexican Mining Law, 12°, Mexico, 1904.

" DR. JAMES DOUGLAS.
Untechnical Addresses on Technical Subjects, 12°, New York, 1904.

" ENGINEERING AND MINING JOURNAL.
152 bound vols., 500 unbound vols., 200 pamphlets, 150 duplicates.

" ENGINEER'S CLUB OF ST. LOUIS.
World's Fair Souvenir, 8°, 1904.

" B. F. FACKENTHAL, JR.
The Thomas Iron Company, 1854–1904, Semi-Centennial Jubilee Volume, ob. 4°, New York, 1904.
Pennsylvania State College—Department of Mining. Mining Bulletin, vol. iv., Nos. 1 and 6, 8°, 1898.
Proceedings of the Special Meeting of Stockholders, June 1, 1904, *to Celebrate the Fiftieth Anniversary of the Formation of The Thomas Iron Company*, 8°, New York, 1904.

" GEOLOGICAL SOCIETY OF AMERICA.
Index to Volumes 1–10, 8°, Rochester, 1900.

" GEORGIA GEOLOGICAL SURVEY.
Bulletin, 11, 8°, Atlanta, 1904.

" U. S. GRANT.
Minnesota Geological and Natural History Survey, Annual Report, xiii.–xv., 8°, St. Paul, 1885–'87.

" GREAT BRITAIN, PATENT OFFICE.
Patents for Inventions. Abridgments of Specifications from 1588–1888. 133 vols., 12°, London, 1857–'89.

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FROM CAPT. ANDREW HERO, JR.

Journal U. S. Artillery, vol. vi.

" L. E. HOWARD.

Offinger's Technical Pocket Dictionary, English, French, Italian, German, 4 vols. (vol. i., ed. 3, vols. ii.–iv., ed. 2), 16°, Stuttgart, n. d.

" INDIANA, DEPARTMENT OF GEOLOGY AND NATURAL RESOURCES.

Annual Report, 28, 8°, Indianapolis, 1904.

" IOWA GEOLOGICAL SURVEY.

Annual Report, 16, 4°, Des Moines, 1904.

" INSTITUTION OF CIVIL ENGINEERS.

Proceedings, vol. clvii., 8°, London, 1904.

" INSTITUTION OF ENGINEERS AND SHIPBUILDERS IN SCOTLAND.

Transactions, vol. xlvii., 8°, Glasgow, 1904.

" INSTITUTION OF MECHANICAL ENGINEERS.

General Index to Proceedings, 1885–1900, 8°, London, n. d.

" INSTITUTION OF NAVAL ARCHITECTS.

Transactions, vol. xlvi., 8°, London, 1904.

" DR. ALBERT R. LEDOUX.

Arte o Nuevo Modo de Beneficiar los Metales de oro y Plata, y de Plata Con ley de Oro, Por Azogue, 12°, Mexico, 1758.

" R. W. LODGE.

Lodge's Notes on Assaying, 8°, New York, 1904.

" W. A. MACLEOD.

Macleod and Walker's Metallurgical Analysis and Assaying, 8°, London, 1903.

" A. MARTINS.

Das Königliche Materialprufüngsamt der Technischen Hochschule Berlin. Denkschrift zur Eroffnung, $^{2}4°$, Berlin, 1904.

" MINNESOTA GEOLOGICAL AND NATURAL HISTORY SURVEY.

Final Report, 1872–1900, 5 vols. in 6, f°, Minneapolis, 1881 1900.

Bulletin No. 1, 8°, St. Paul, 1889.

" JAMES PARK.

Park's Cyanide Process, 12°, London, 1900.

" PRINCETON COLLEGE LIBRARY.

Subject Catalogue of the Library of the College of New Jersey, nar. 4°, New York, 1884.

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FROM R. W. RAYMOND.

100 bound vols., 175 pamphlets, 28 duplicates, 1 map.

" RHODESIA CHAMBER OF MINES.

Annual Report, 9, 8°, Bulawayo, 1904.

" C. H. SNOW.

Hutton's Principles of Stone Bridges. Title page wanting. 8°, 1772.

" J. M. SWANK.

American Iron and Steel Association. *Directory to the Iron and Steel Works of the United States*, edition 16, 1904. (2 copies.)

Supplement to the Directory, 1903.

Swank's Notes and Comments on Industrial, Economic, Political and Historical Subjects, 12°, Philadelphia, 1897.

" TASMANIA, SECRETARY FOR MINES.

Report, 1903, 8°, Hobart, 1904.

Report on the Zeehan Silver-Lead Mining Field, 8°, Hobart, 1904.

" U. S. GEOLOGICAL SURVEY.

Annual Report, vol. xxii., pts. 3–4, 4°, Washington, 1902. *Professional Papers*, Nos. 21–24, 26–28, ²4°, Washington, 1904. *Topographic and Geologic Index Map*, Feb., 1904.

" U. S., LIBRARY OF CONGRESS.

A. L. A. Catalogue, 1904, 8°, Washington, 1904.

" U. S., PATENT OFFICE.

Catalogue of Additions to the Library, 1878–'83, 4°, Washington, 1883.

" U. S. BUREAU OF STATISTICS.

Foreign Commerce and Navigation in the United States, 1903.

" WESTERN AUSTRALIA DEPARTMENT OF MINES.

Reports, 1895–1902, f°, Perth, 1896–1903.

" WESTERN AUSTRALIA GEOLOGICAL SURVEY.

Annual Progress Report, 1902, f°, Perth, 1903. *Bulletin No.* 10, 8°, Perth, 1903.

PURCHASES.

American Chemical Journal, vols. i.–vii., 8°, Baltimore, 1879–'85.

Chemical News, vols. xi.–xxvi., xxix.–xxxi., xxxvii.–xlv., 8°, London, 1865–'82.

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CONGRÉS GÉOLOGIQUE INTERNATIONAL. *Compte Rendu*, Sessions 4 and 9, pts. 1–2, 8°, 1891 and 1904.

International Encyclopædia, vols. i. to xx., 8°, New York, 1904.

IRON AND STEEL INSTITUTE. *Transactions*, 1869–1870.

——— *Journal*, 1871–1890. Wanting: 1871, pt. 1, pp. 1–276.

EDWARD KIRK. *Founding of Metals*, 8°, New York, 1885.

Mechanics' Magazine, vols. i.–vi., 8°, London, 1824–'28.

Mining Magazine, vol. iv., 8°, New York, 1855.

HENRI MOISSAN. *Electric Furnace.* Translated by A. T. de Mouilpied. 8°, London, 1904.

NEWCASTLE-UPON-TYNE CHEMICAL SOCIETY. *Transactions*, vols. i.–v., 8°, 1871–'83. All published?

NEW YORK STATE GEOLOGIST. *Annual Report*, 12, 14, 16–19, 8°, Albany, 1893–1901.

Quarterly Journal of Science, vols. iv.–x., 8°, London, 1867–'73.

SOCIETY OF ENGINEERS. *Transactions*, 1861–1902, 8°, London, 1861–1903.

U. S. ORDNANCE DEPARTMENT. *Report of the Tests of Metals*, 1882, '83, '85, '88–'92, '96.

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MEMBERSHIP.

The following list comprises those persons elected as members who duly accepted election in November and December, 1904.

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GASTON STRAP, . . . 24 Rue Lalo, Paris, France.
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CHARLES S. THOMAS, . . P. O. Box 1117, Ouray, Colo.
SAMUEL WHITWORTH, Glebe Road Works, Kingsland Rd., London, N. E., Eng.

The names of one hundred and seven applicants for membership were sent out for postal ballot December 10, 1904. All these applicants will be declared elected January 10, 1905, and the names of those duly accepting before the time of going to press will be given in Bi-Monthly Bulletin No. 2, for March, 1905.

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SECTION II.

TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents, pages i and ii.

The Proceedings of the 87th Meeting (pp. 181–236) will be condensed for publication in the volume of *Transactions*, the names of local committees, list of papers and records of technical sessions being retained. Those who desire to possess the detailed account of excursions, etc., should preserve this number of the Bulletin.

Errata.

The following corrections and additions to papers contained in this Section were received too late for incorporation in the text of said papers; but will be introduced in their proper places in the volume of the *Transactions*.

Page 21, line 2; and p. 22, title of Fig. 10, add "(the record of each day is averaged with those of all preceding days)."

Page 21, line 8 from bottom. For "gals." read "U. S. standard gallons."

Pages 175 and 177. The map of the Long-Wall Workings at the Valley Mine should be Fig. 3 and appear on p. 177, while the cut now on p. 177 should appear on p. 175 as Fig. 2, with the title "Room-and-Pillar Working and Open-Pit Mining at Minden, Barton County."

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The Application of Dry-Air Blast to the Manufacture of Iron.

BY JAMES GAYLEY, NEW YORK, N. Y.

(Lake Superior Meeting, September, 1904.)*

THE atmosphere, which plays such an important part in the manufacture of iron and steel, is the most variable element involved in its several processes; and particularly is this true of the blast-furnace process, which consumes air in large quantities. At no time since the blast-furnace became an important and widely used apparatus—even when it was operated in the most crude manner—have the variations in composition of the raw materials used been as frequent and as great as the variations in humidity of the atmosphere. Important improvements have been made in the blast-furnace and its accessories, such as the hot-blast stoves, the increase in size and change in the shape of the furnace, more efficient blowing-engines, the increased protection given to the bosh walls, and the careful preparation of the raw material, all of which have exerted a pronounced influence on the furnace-operations from a metallurgical standpoint. But during the past eight years little advance has been made in this direction; the fuel-consumption has not diminished, nor has there been any material increase in production. Within that period, however, there has been witnessed the greatest development in appliances for the economical handling of material. So complete has been the work in this direction, that, except in isolated cases, in this country at least, a further extension does not hold out much promise of a satisfactory return on the investment required. It seemed that, with the exception of the gas-engine, we had about reached the limit, for like a strong wall, the atmosphere, with its humidity as variable to-day as when first blown into a primitive blast-furnace, appeared to stand as a barrier to

* Read by title at the Lake Superior meeting, but first presented at the New York meeting of the Iron and Steel Institute (October, 1904), and published under a mutual agreement in the *Transactions* of both Institutes.

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further progress. In furnaces using ore from the Lake Superior district the raw material, amounting to about 7,200 lb. per ton of iron, varies in composition within 10 per cent. and is as uniform as human skill can make it; but the atmosphere, of which 11,700 lb. are consumed per ton of iron, varies in its content of moisture from 20 to 100 per cent., from day to day and often in the same day, thus rendering the process, even with the best appliances, an uncertain one, because dependent on the caprice of the atmosphere.

The desiccation of the air used in blast-furnaces in such a way as to reduce its moisture to a small quantity, and to keep it uniform, must of necessity contribute in a very marked degree toward the attainment of uniformity in the furnace-operations. The advantages from desiccation can be appreciated only after due consideration is given to the volume of air that is consumed per minute and the large amount of moisture which it contains. Managers of blast-furnaces are familiar with the chilling-effects produced in the hearth by a tuyere that is leaking, which immediately results in a deterioration in the grade of the iron; yet the quantity of water ordinarily entering the furnace under these conditions is not greatly in excess of the quantity carried in, like a steady stream, by the atmosphere, during a period of the average humid conditions prevailing in the summer-season in this country.

It has been deemed preferable in this communication to express the quantity of moisture, contained in the atmosphere as aqueous vapor, in grains of water per cubic foot of air, inasmuch as the quantity of air blown into blast-furnaces is expressed in cu. ft. With air containing 1 grain of water per cu. ft., there is passed into the furnace, for each 1,000 cu. ft. used per min., practically 1 gal. of water per hour. A furnace of average size in the Pittsburg district consumes about 40,000 cu. ft. of air per minute, which would pass into the furnace 40 gal. of water per hour for each grain of moisture contained in a cu. ft. of air. The quantity of moisture in the air, taken from daily readings by the observer of the United States Weather Bureau at Pittsburg, is set forth in Table I.

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TABLE I.—*Moisture in Air at Pittsburg. U. S. Weather Bureau.*

	Average Temperature.	Weight of Water per Cu. Ft. of Air.	Quantity of Water Entering per Hour into a Furnace Using 40,000 Cu. Ft. of Air per Minute.
	Degrees Fahrenheit.	Grains.	Gallons.
January,	37.0	2.18	87.2
February,	31.7	1.83	73.2
March,	47.0	3.40	136.0
April,	51.0	3.00	120.0
May,	61.6	4.80	192.0
June,	71.6	5.94	237.6
July,	76.2	5.60	224.0
August,	73.6	5.16	206.4
September,	70.4	5.68	227.2
October,	56.4	4.00	160.0
November,	40.4	2.35	94.0
December,	36.6	2.25	90.0

Table I., like all records made by the U. S. Weather Bureau, is from observations taken on the top of a high building, and does not correctly indicate the condition of the atmosphere at the furnaces where the air is used. In fact, at one of the steel-works in Pittsburg, observations made simultaneously at three separate stations showed a perceptible variation in moisture. For the purpose of comparison with observations of the U. S. Weather Bureau, there is shown in Table II. the average monthly content of moisture in the air at the furnaces, the observations being made at 9 a.m.

TABLE II.—*Moisture in Air at Steel-Works in Pittsburg.*

	Weight of Water per Cu. Ft. of Air.
	Grains.
January,	2.8
February,	2.7
March,	3.1
April,	3.3
May,	4.7
June,	7.3
July,	7.0
August,	7.1
September,	5.4
October,	3.2
November,	3.3
December,	3.0

The variations in moisture from month to month set forth clearly the conditions, as to atmosphere, with which blast-fur-

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naces in this country have had to contend. If these conditions were uniform throughout the whole month, it would not be a difficult problem to deal with; but unfortunately they are not uniform, and it is instructive to note the changes which occur from day to day in the same month. In Table III. is shown a record worked out from data furnished by the U. S. Weather Bureau at Pittsburg. These observations represent a different period from that shown in Table I.; they were taken at 8 a.m. and 8 p.m., and show the grains of water per cubic foot of air at the time observed, for the months of January and July.

TABLE III.—*Moisture in Air at 8 a.m. and 8 p.m.*

	JANUARY. Quantity of Water.		JULY. Quantity of Water.	
	8 A.M.	8 P.M.	8 A.M.	8 P.M.
Day of Month.	Grains.	Grains.	Grains.	Grains.
1,	1.96	3.06	7.24	7.48
2,	2.55	3.66	8.23	7.98
3,	2.46	3.80	8.50	7.48
4,	2.07	2.27	8.50	7.48
5,	1.81	1.12	8.46	7.72
6,	0.99	1.12	6.50	8.24
7,	1.16	1.67	8.78	7.47
8,	1.49	1.88	7.98	7.24
9,	1.96	2.19	6.78	5.94
10,	1.81	1.88	7.48	6.35
11,	1.74	1.55	7.98	7.48
12,	1.55	1.07	6.73	6.35
13,	0.99	1.55	5.94	4.84
14,	1.61	1.81	5.55	5.74
15,	1.67	1.96	5.74	5.19
16,	2.04	2.27	6.35	6.35
17,	2.45	3.29	7.72	7.98
18,	1.81	1.32	7.24	7.24
19,	1.12	1.16	8.24	7.48
20,	1.43	2.11	7.48	7.24
21,	2.11	1.88	7.72	7.38
22,	1.88	1.88	6.78	5.74
23,	0.91	1.17	7.43	6.35
24,	0.99	2.11	6.56	6.11
25,	0.69	1.83	6.05	7.74
26,	0.61	0.99	7.72	7.32
27,	0.56	0.88	7.98	7.48
28,	0.72	0.70	6.56	5.74
29,	0.76	0.80	6.14	5.01
30,	0.95	1.12	5.74	6.35
31,	0.70	1.41	6.56	5.19

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It will be observed from the data in Table III., that, while the moisture in the atmosphere in the month of January is much less than in July, yet the percentage of variation is greater. In order to illustrate more precisely the exact conditions with respect to the atmosphere, under which blast-furnaces must be operated, there is shown in Tables IV. and V. a record of observations taken each hour in the day. In order not to make the data too burdensome the months of April and October have been selected, as they represent months between the warm and cold seasons, and will also serve for comparison with January and July, as shown in Table III.

It should be stated with reference to the Tables IV. and V., that observations were taken with a stationary instrument, which shows results somewhat higher, and not as accurate, as those taken with a whirled psychrometer. Nevertheless, they were taken with the same instrument and are relatively correct. By simply multiplying the grains of moisture by 40—which represents the number of gallons of water entering a modern furnace per hour, for a content of 1 grain of moisture in a cubic foot of air—a clear idea can be had of the gallons of water entering the furnace per hour, for the various conditions of humidity. The changes are great not only from day to day, but from hour to hour in the same day, and often they are very abrupt. These records were made at a furnace-plant, situated on thebank of a river, where the conditions exist for an increased humidity as compared with higher ground; and to what extent the abrupt changes may have been caused by the presence of steam in the atmosphere—absorbed from spraying of the hot pig-beds, the blow-off from boilers and exhaust from engines, or from a rain-storm, when the humidity decreases suddenly—it is impossible to say. How frequently has it happened in the experience of every furnace-manager, that the furnace has gradually or suddenly lost its hearth-temperature and produced a grade of iron either undesirable or unmarketable, without any visible cause. Tuyeres are examined for leaks, the raw material in the stock-yard is carefully inspected, and usually the coke is condemned. A more intimate acquaintance with the atmosphere would have provided a correct and ready reason, for the variations therein are not only many times greater than in the raw material, but a greater weight of it is used per ton of iron.

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TABLE IV.—*Hourly Record of Moisture in Air.*

Grains of moisture per cubic foot of air.

April	A.M.							P.M.												A.M.				
	6	7	8	9	10	11	12	1	2	3	4	5	6	7	8	9	10	11	12	1	2	3	4	5
1	2.77	3.30	3.71	3.71	3.09	3.46	3.46	3.46	3.46	3.46	3.46	4.11	3.71	3.71	3.71	3.30	3.71	3.71	3.71	3.30	3.71	3.71	3.71	3.09
2	3.71	3.71	3.30	3.30	3.71	3.71	3.46	3.46	3.46	3.81	3.46	3.81	3.09	3.81	3.30	3.77	2.77	3.71	3.71	3.71	3.71	3.71	3.68	3.30
3	3.09	3.09	3.09	3.09	3.46	3.46	3.46	3.09	3.09	3.46	3.09	3.30	3.30	3.02	3.02	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30	3.30
4	3.71	3.71	3.71	3.71	3.71	4.11	4.11	4.67	4.67	3.81	3.81	4.11	3.30	4.01	3.30	3.71	4.01	3.71	3.71	3.30	3.38	3.30	3.30	4.01
5	3.68	3.68	4.01	3.30	4.01	3.89	3.71	4.11	4.11	3.46	4.11	4.11	4.11	3.46	3.09	3.09	3.09	2.47	2.77	2.77	2.77	2.77	2.77	2.77
6	2.47	3.09	3.09	3.09	3.71	3.71	3.71	3.71	3.46	3.46	3.71	3.71	3.71	3.71	3.71	3.09	3.30	3.30	3.30	3.30	2.77	2.77	2.77	2.47
7	2.77	3.30	3.30	2.77	4.01	3.30	3.71	3.81	4.11	4.33	4.67	4.67	4.11	4.01	3.30	3.30	3.71	3.71	3.30	3.30	3.30	3.30	3.30	3.30
8	3.02	3.30	4.01	3.30	3.71	3.71	3.71	4.11	4.11	4.40	4.40	4.40	3.71	3.71	4.01	4.01	3.30	3.30	3.30	3.30	3.30	3.02	3.02	3.30
9	3.68	3.68	3.71	4.11	4.11	4.11	4.28	4.67	4.01	4.01	4.31	4.31	3.68	3.68	3.68	4.01	3.68	3.68	3.30	3.39	3.39	3.02	3.39	3.02
10	3.68	3.68	3.98	3.68	3.68	4.01	3.71	3.71	4.67	4.17	5.11	5.46	5.11	5.35	5.11	5.14	5.35	5.20	4.75	4.40	4.75	4.40	4.75	4.75
11	4.75	4.31	4.40	5.01	4.33	5.28	4.56	4.56	5.22	5.22	5.22	5.22	4.24	4.96	3.81	3.81	3.46	3.46	3.81	3.81	3.46	3.46	3.46	3.46
12	3.46	3.46	3.30	3.46	3.98	3.30	3.84	3.68	3.47	3.20	3.52	3.52	3.17	3.11	2.74	3.98	3.46	3.71	3.71	3.09	3.09	3.30	3.30	3.30
13	3.02	3.68	3.71	4.01	4.96	4.71	5.22	4.85	4.59	4.85	4.56	4.56	4.56	4.96	4.96	4.67	4.81	5.01	5.01	4.75	4.75	4.31	4.31	4.31
14	3.68	3.68	3.68	3.68	4.31	4.01	4.01	4.01	4.01	4.31	4.31	4.40	4.31	4.31	4.31	4.31	4.31	3.98	4.31	3.68	3.98	4.31	3.98	3.98
15	4.01	4.01	3.71	4.40	4.41	4.75	4.33	4.96	4.56	4.24	4.24	4.24	4.33	3.81	4.11	3.81	3.46	3.30	3.30	3.30	3.30	3.68	3.62	3.62
16	3.68	3.68	4.01	3.81	3.81	4.67	3.52	4.96	4.96	4.96	4.96	4.96	4.96	5.01	4.40	4.40	4.11	4.75	4.46	4.67	5.01	5.01	5.01	5.01
17	5.01	4.67	5.60	5.26	5.28	5.97	6.27	6.27	6.27	6.27	5.83	6.27	6.25	5.60	5.60	5.60	5.35	5.35	5.77	4.40	4.75	4.75	4.75	4.52
18	4.75	4.75	5.35	5.35	5.35	5.32	5.60	6.08	6.43	6.43	6.62	5.84	5.84	5.84	5.77	5.77	5.11	5.11	5.77	5.77	5.77	5.35	5.77	5.77
19	5.77	5.35	5.77	5.77	6.32	5.77	6.32	6.32	6.32	6.32	6.32	5.84	6.72	5.77	4.75	4.75	4.75	4.01	4.01	4.31	3.30	3.30	3.30	4.01
20	4.01	3.30	3.30	3.30	3.30	3.30	3.30	3.71	4.40	4.40	4.40	4.40	3.71	3.30	3.30	3.30	3.30	3.30	3.09	3.30	3.30	3.30	3.30	3.30
21	4.01	4.40	4.40	3.71	3.71	4.40	4.11	4.11	4.11	4.67	5.01	5.01	5.35	4.01	4.01	4.01	5.11	4.75	4.31	4.01	4.01	4.31	5.11	4.31
22	3.30	4.01	3.46	3.71	4.11	4.11	4.33	4.56	4.98	5.22	5.59	5.10	4.24	4.24	4.56	4.24	5.22	5.22	5.22	5.59	5.59	5.28	5.01	4.40
23	5.35	5.01	5.01	5.60	5.28	5.28	5.84	5.35	6.18	6.18	6.18	6.18	6.18	6.39	6.39	6.18	6.18	6.18	5.77	6.18	6.18	6.18	5.11	5.11
24	5.46	5.11	5.11	5.11	5.11	5.11	5.11	5.11	4.75	5.11	4.75	5.77	4.75	5.11	5.11	5.46	5.77	5.77	5.35	4.75	4.75	5.11	5.11	4.75
25	4.31	4.41	4.75	4.75	4.40	4.40	4.40	5.01	5.01	5.01	5.01	5.01	4.75	4.75	4.40	4.40	3.71	3.75	4.05	4.00	4.01	4.01	4.01	4.01
26	4.01	4.40	4.11	4.11	4.61	4.67	4.67	4.67	4.67	4.67	4.33	4.33	4.33	4.33	3.71	4.01	4.31	4.01	3.68	4.01	3.30	3.68	3.68	3.68
27	4.31	4.01	4.40	4.11	4.67	4.67	4.96	4.33	4.33	5.28	4.33	4.96	4.67	3.71	4.01	4.40	4.01	4.01	4.01	4.01	3.68	3.68	3.68	3.68
28	3.30	4.01	3.46	3.81	4.11	4.11	4.67	4.67	4.67	5.28	5.28	4.33	5.01	4.40	4.11	4.40	4.40	4.40	3.71	4.31	4.01	4.01	3.71	4.40
29	4.01	4.01	4.40	4.40	4.40	4.11	4.11	3.81	4.40	4.40	4.11	4.40	4.40	4.01	4.31	4.75	4.75	4.07	4.01	4.01	4.01	4.01	4.31	4.01
30	4.01	3.68	4.31	3.81	4.67	4.33	5.60	5.28	5.97	6.43	5.28	5.28	5.84	5.01	5.35	5.77	4.75	4.75	4.75	5.11	4.75	4.75	4.75	4.75

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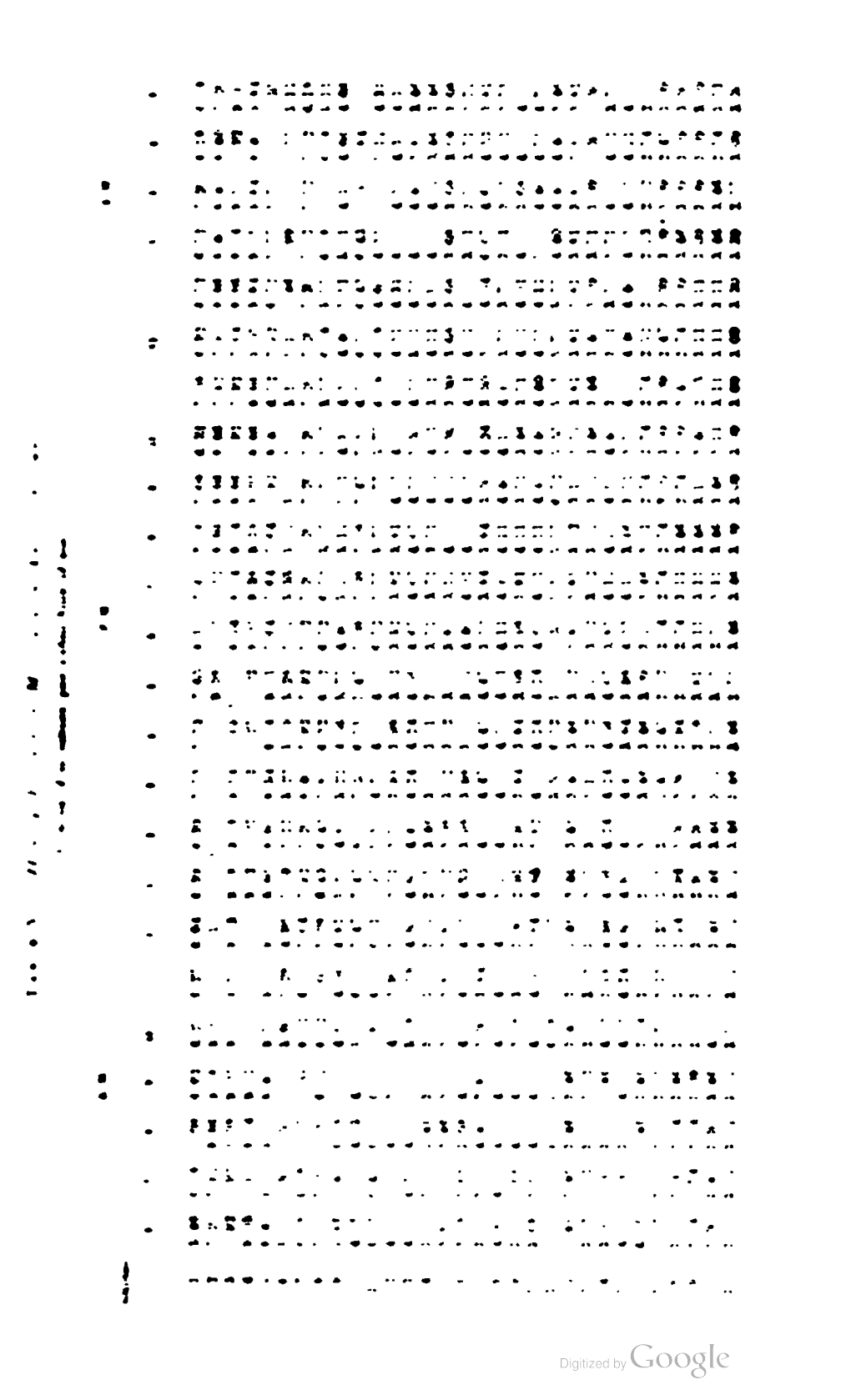

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It is true that the atmosphere has been recognized by numerous metallurgists as the cause of many serious irregularities in blast-furnace operations, but it is doubtful whether its influence has been adequately recognized. Many writers on metallurgical subjects have considered the moisture in the atmosphere, and calculated—and invariably underestimated—the absorption of heat necessary for its dissipation; and have dismissed the subject with the conclusion that to extract the moisture the game was not worth the candle, or in a spirit of resignation accepted it—like storm and sunshine—as a condition beyond our control. This conclusion has no doubt been reached by a consideration alone of the quantity of fuel necessary to dissipate the moisture in the furnace hearth, based on observations of the humidity of the atmosphere taken outside the blowing-engine room; and this quantity, while important, does not indicate a great saving in fuel. Of much greater importance is the variation in moisture from time to time and the margin of heat carried in the furnace to compensate for these variations, which margin is invariably large; and every furnace manager is aware of its existence, from the way in which he is required to manipulate the hot-blast temperatures, and from the silicon in the metal, which is the thermometer of the hearth.

It has often been a matter of surprise that a greater saving of fuel per ton of iron was not obtained in winter, as compared with summer season, since the winter records show much less moisture in the atmosphere. The reason is that blowing-engines at blast-furnaces do not receive air of the dryness shown in the tables above. In summer the windows and doors of the blowing-engine room are wide open, and the humidity of the air-supply is practically that of the atmosphere; but in winter they are nearly or quite closed, and the entering-air has mixed with it all of the steam that leaks from the engine. Records taken over a number of years show that there is not a very great difference in the moisture in atmosphere between observations taken outdoors in summer and in the engine-room in winter. In Table VI. are monthly records showing a comparison between winter and summer months, the observations having been taken indoors and outdoors respectively.

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TABLE VI.—*Moisture in Air in Winter and in Summer.*

WINTER.		SUMMER.	
Month.	Quantity of Water per Cu. Ft. of Air.	Month.	Quantity of Water per Cu. Ft. of Air.
	Grains.		Grains.
January, . . .	4.5	April, ...	4.2
February, . . .	4.6	May, ...	4.1
March, . . .	4.7	June, ...	6.4
October, . . .	6.4	July, ...	5.2
November, . . .	4.6	August, ...	6.7
December, . . .	5.0	September, . . .	5.7

From a comparison of Table VI. with Table II. it certainly appears that a material advantage could be gained by leading pipes from outdoors to the inlet-valves of the air-cylinder. So impressed did I become with this conclusion, that the blowing-engines at a furnace under my direction were thus equipped in the month of January, and continued to draw the supply of air from outdoors throughout the year. The excellent results expected in the winter season did not materialize, or rather, were so slight, as compared with a companion furnace not so equipped, as to argue against any further experiment in that line. This experience suggested the conclusion that while the air in the engine-room was higher in its content of moisture, through its admixture with steam, than the outside air, yet it was not subject to the same variations; and further that these variations, which were often sudden and great, were really the most troublesome feature, and that nothing less than maintaining the atmosphere uniform with respect to humidity would prove of any material advantage. The saving in fuel through such uniformity could not be accurately set forth. The amount of fuel necessary for the decomposition of the moisture in the blast could be closely arrived at; but to what extent the reserve of heat, needed for counteracting the variations in moisture, could be diminished by desiccation of the blast, could only be approximately estimated, in the absence of definite data. Nevertheless, the possible saving in that respect was deemed to be considerable.

Of course, a wide field of experiment had to be covered in order to determine the most feasible method and apparatus for extracting the moisture. Various schemes for its direct absorp-

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tion were worked out and in turn abandoned; and refrigeration by means of anhydrous ammonia was finally chosen. After many preliminary experiments an insulated chamber containing coils of pipe, and of sufficient size to treat the air from a blowing-cylinder 3 ft. in diameter, was built. A small ice-machine was installed to circulate the ammonia through the coils, and the air was admitted to the refrigerating-chamber from an auxiliary chamber into which steam could be introduced at will, thus making it possible to treat, at any time, air containing the maximum amount of moisture with which it would be necessary to contend in the summer months. In this experimental plant air was treated under a variety of conditions for a considerable period, and from the data obtained the equipment for a modern furnace was worked out.

In the original plan, the refrigerating-chamber was placed between the blowing-engine and the furnace, so that the air passed through it under blast-pressure. At a furnace of modern type, this pressure may range from 15 to 30 lb. per sq. in. —more than double that which was in use when this process was first conceived. In the practical working-out of the process, it was found that workmen must frequently enter the refrigerating-chamber, to regulate the flow of the refrigerant through the pipes, and the thawing-off of the frost. A high pressure in the chamber would make its supervision and operation extremely difficult and hazardous. This consideration, together with that of the more expensive construction required by the high pressure of the passing air, led to the location of the chamber so that the air should be drawn through it at atmospheric pressure to the blowing-engines.

The Isabella furnaces of the Carnegie Steel Co., located at Etna, Pa., a suburb of Pittsburg, were selected as the plant at which to install the apparatus for applying the dry-air blast.

The lines and dimensions of this furnace, shown in Fig. 1, represent the usual construction of furnaces in the Pittsburg district. The furnace is blown with twelve 6-inch tuyeres, and is equipped with four hot-blast stoves. Blast is supplied by three blowing-engines having the following dimensions: steam-cylinder, 44 in. in diameter; air-cylinder, 84 in.; stroke, 60 inches.

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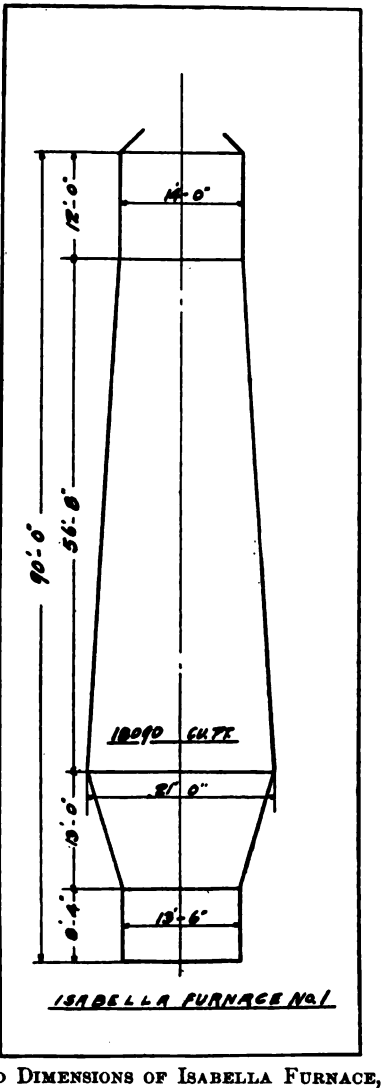
Fig. 2 shows in elevation the ammonia-compressors, condensers, and the refrigerating-chamber. This view of the re-

FIG. 1.



LINES AND DIMENSIONS OF ISABELLA FURNACE, No. 1.

frigerating-chamber shows it to be connected for the direct expansion of ammonia; but, as the escape of ammonia gas through a broken pipe or leaking-joint might imperil the life of anyone

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in the chamber at the time, it was decided to adopt the brine system, and the pipe-connections are as shown in Figs. 3 and 4, representing the refrigerating-chamber in end-view and ver-

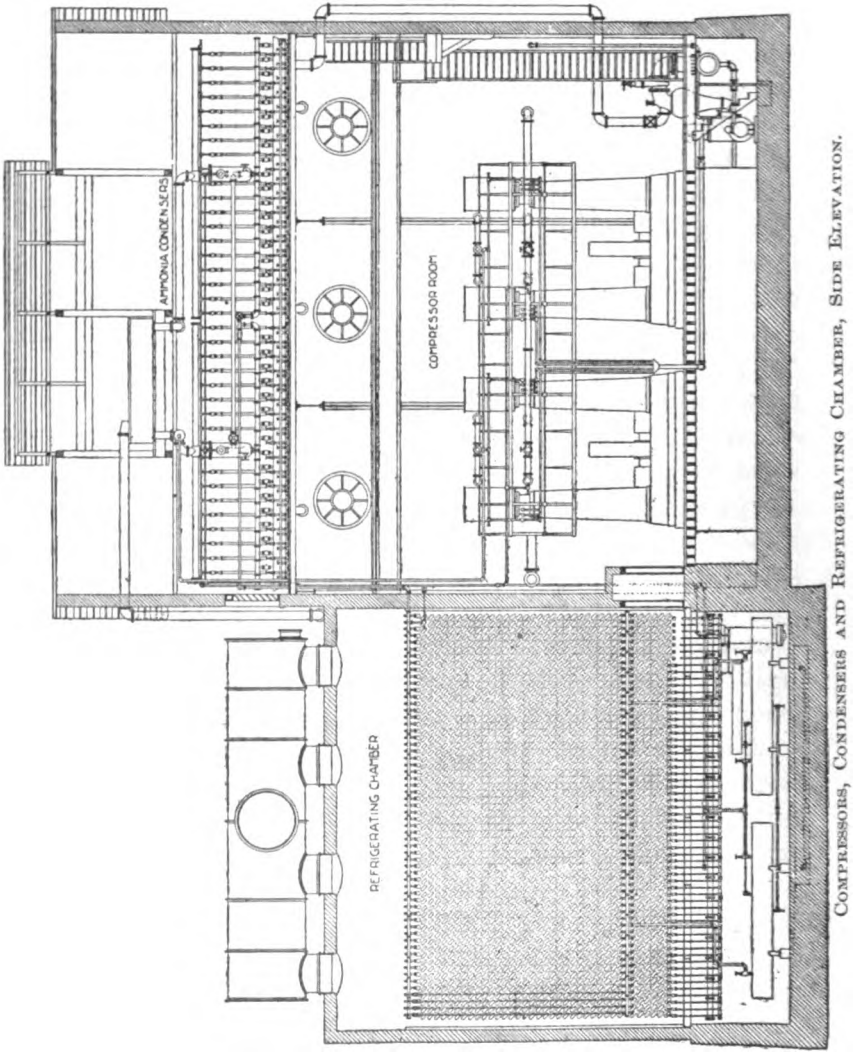

FIG. 2. COMPRESSORS, CONDENSERS AND REFRIGERATING CHAMBER, SIDE ELEVATION.

tical section. The refrigerating-chamber is lined on the inside with plates of compressed cork 2 in. thick.

The ammonia-machines are of the compressor type, and were built by the York Manufacturing Co., York, Pa. The dimensions are as follows: Diameter of the high-pressure steam-

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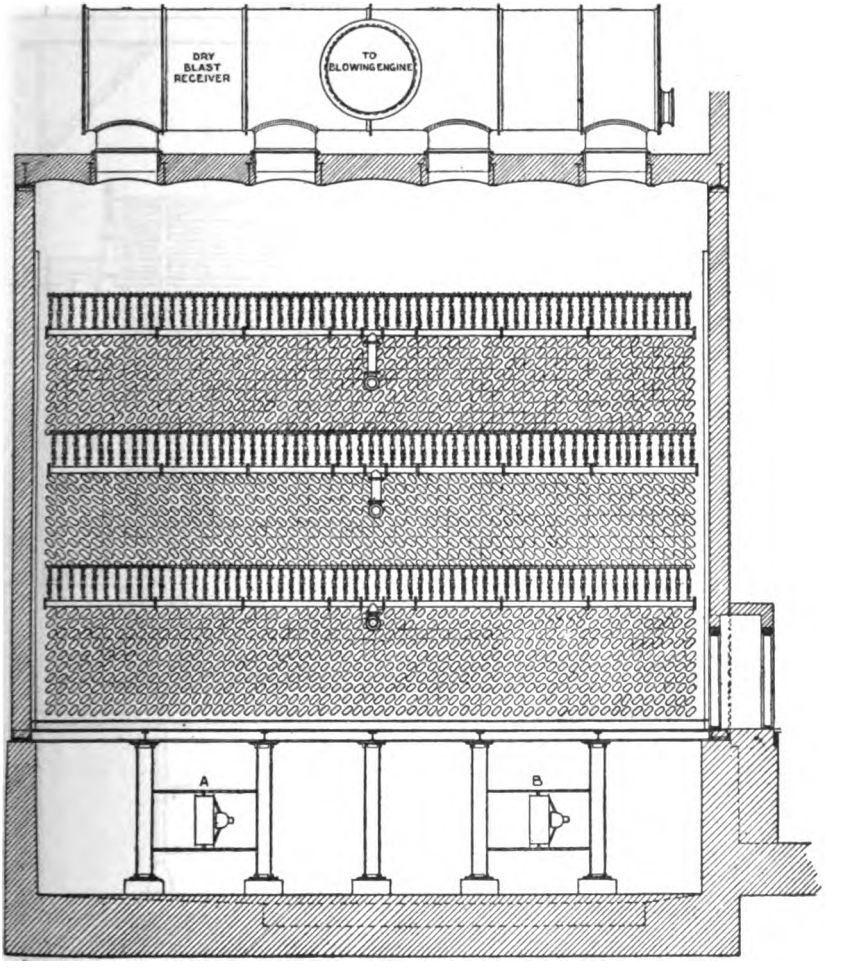
cylinder, 28.5 in.; low-pressure, 56 in.; compressor cylinder, 22.5 in.; stroke, 36 in. Two compressors were installed in order to have one in reserve at all times, as a funace operating on uniformly dry air cannot be subjected to ordinary atmospheric

FIG. 3.



REFRIGERATING CHAMBER, TRANSVERSE SECTION.

conditions without serious results, and frequently on very humid days the assistance of the second engine might be required. Each compressor has the capacity to melt 225 tons of ice.

Fig. 5 shows the brine-tank, containing 20 coils of pipe of

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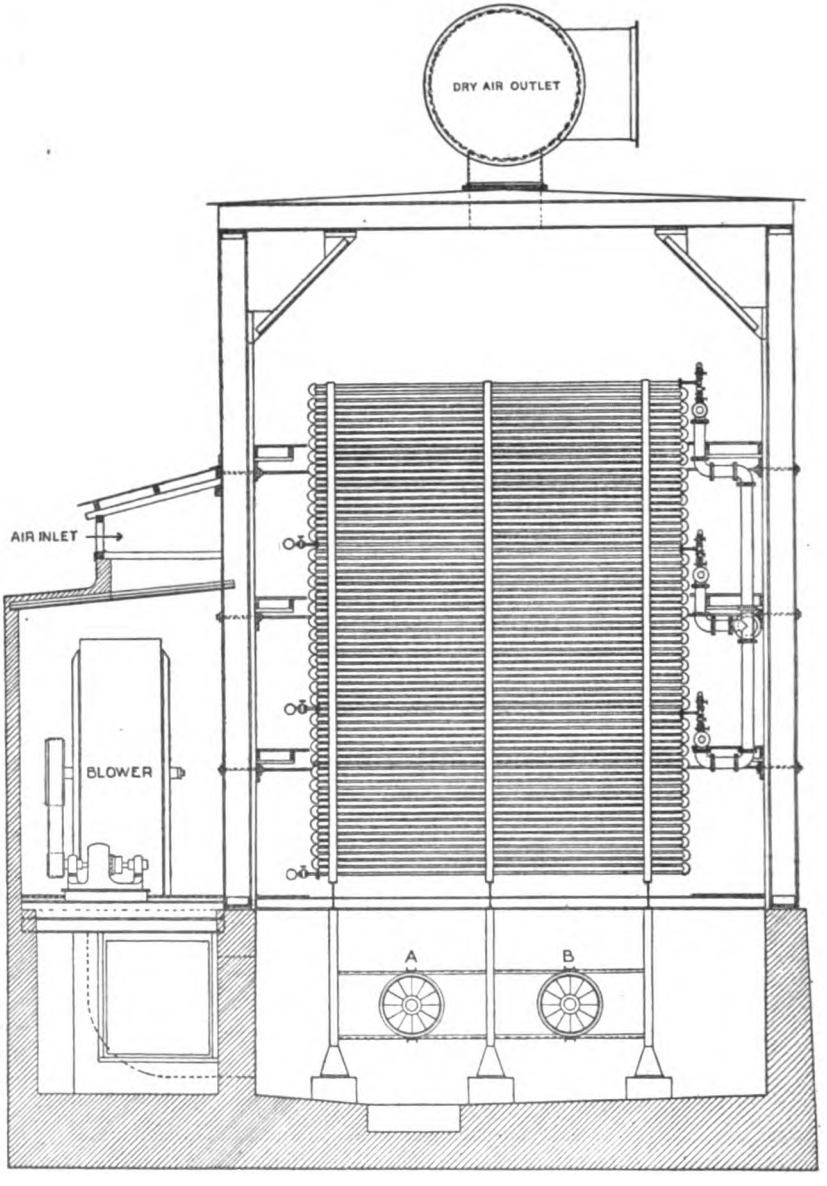
FIG. 4.



REFRIGERATING CHAMBER, LONGITUDINAL SECTION.

the dimensions shown in the diagram. The coils are covered with calcium chloride brine having a sp. gr. of 1.21. The return brine from the refrigerating-chamber flows into the

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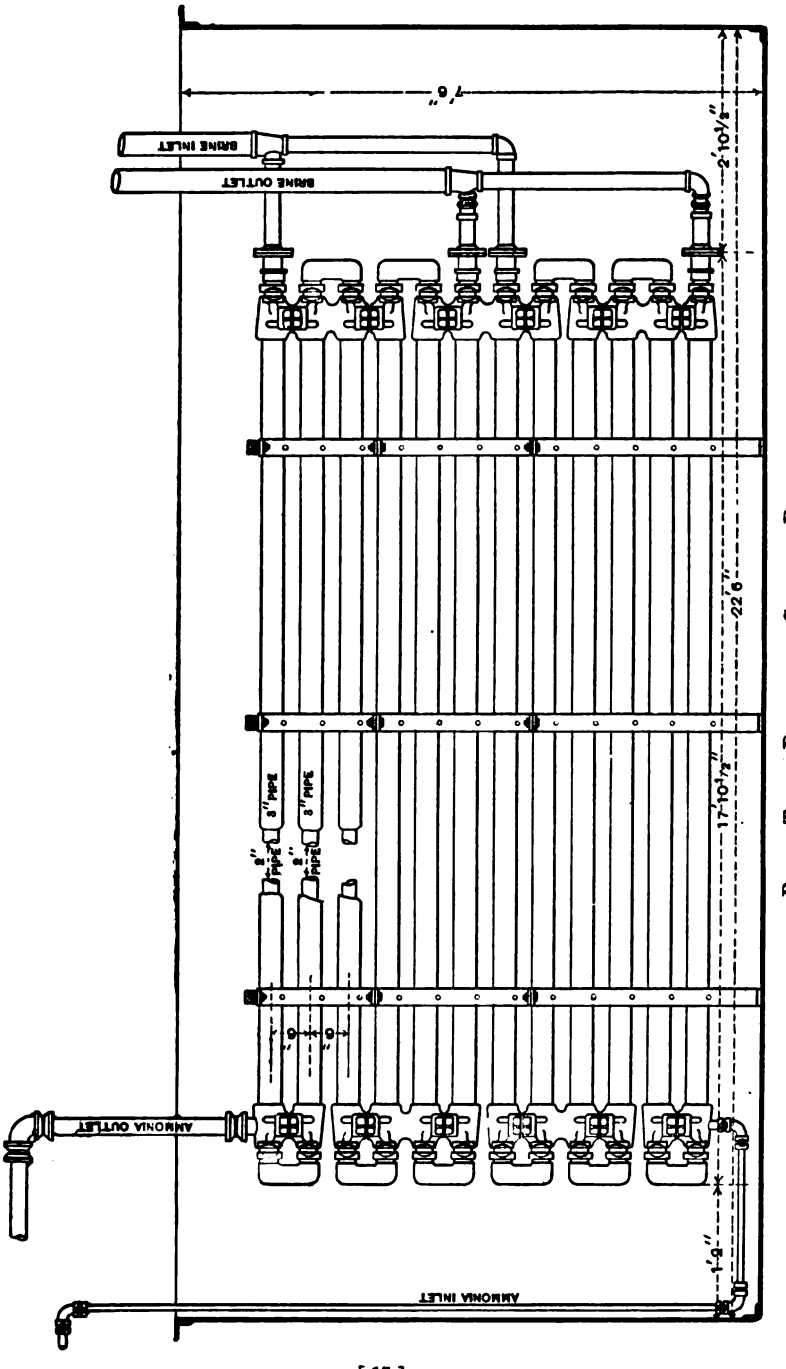

FIG. 6.

BRINE TANK, DETAILS OF COILS OF PIPES.

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top of the tank, is cooled by the ammonia expanding between the outer and inner pipes, withdrawn by a pump, and forced back through the pipe marked "brine-inlet" into the 2-in. or inner pipes (where it is cooled below the freezing-point), and thence into the coils in the refrigerating-chamber. The ammonia enters at the bottom of the pipes, thus traveling in the opposite direction from the brine, and by expanding between the 2-in. and 3-in. pipe, cools the brine, both in the tank and in the inner pipes; 40,000 gal. of brine are required in the system.

Figs. 3 and 4 show the arrangment of pipes in the refrigerating-chamber. There are in each vertical line of coils seventy-five 2-inch pipes 20 ft. long, and in the chamber sixty such vertical lines, the whole representing 90,000 linear feet of 2-in. pipe in the chamber. The pipes in each vertical coil are placed in staggered position to insure better contact with the air. The series of coils is divided into three sections, each fed through a 4-in. header, and discharging into a 6-in. header and thence into a standpipe, from which the brine flows to the brine-tank, the feed being arranged so that the brine flows in a direction opposite to that of the air. As the space between the pipes would become gradually reduced through the accumulation of frost, which might diminish the efficiency of the blowing-engine, a blower was installed to force air into the refrigerating-chamber; and in order to secure a uniform distribution of air over the coils, revolving-fans (A and B, Figs. 3 and 4) were placed in the space underneath, so that all the coils would frost alike. The entering air, according to its humidity, deposits its moisture in the form of water or frost on the lower pipes and as frost only on the upper pipes, and passes from the top of the chamber to the blowing-engines at or below the temperature of freezing, and with a practically uniform content of moisture. When the pipes have become covered with frost the cold brine is shut off from several vertical lines of coil at once; by means of an auxiliary pump and line of pipe, brine that has been heated in a tank with steam is forced through; and in a few minutes the frost is melted. Connection is then made with the cold-brine system, and frost begins to deposit quickly. The frost which has been melted off the pipes collects in a trough in the basement floor, from which it flows into the supply-tank for the condenser.

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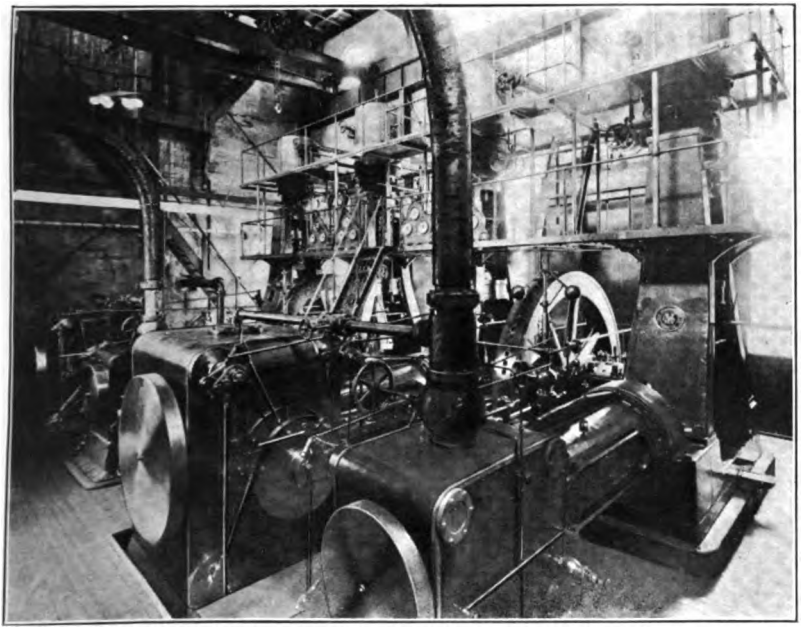
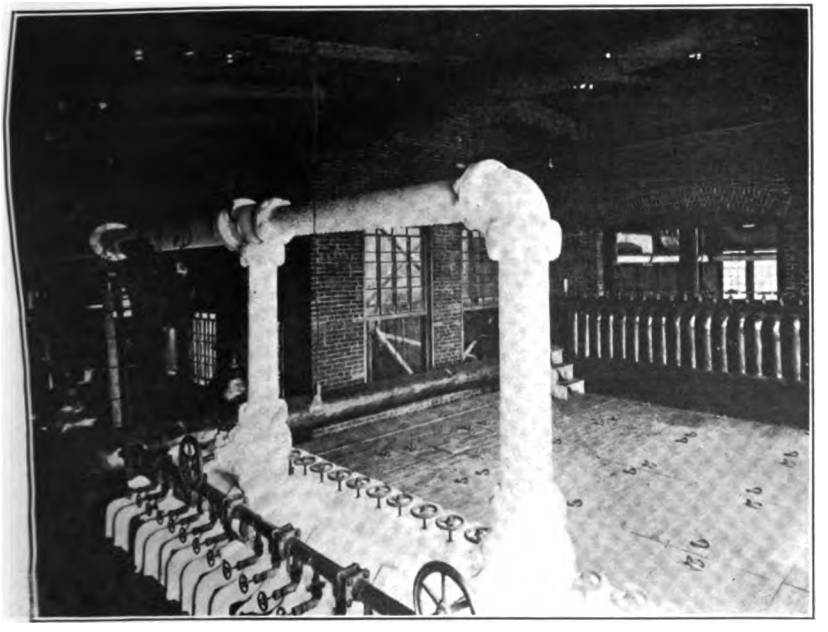
FIG. 6.



AMMONIA COMPRESSORS.

FIG. 7.



BRINE COOLING-TANK.

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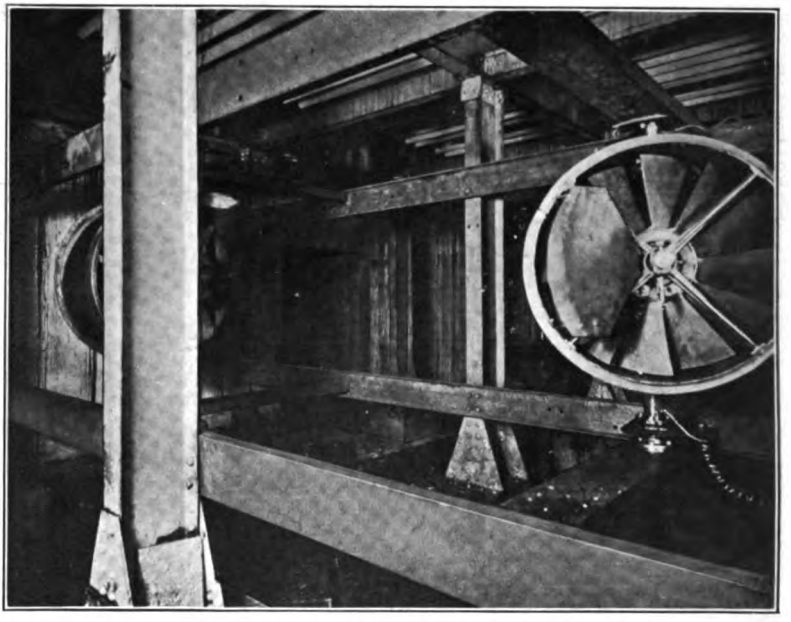
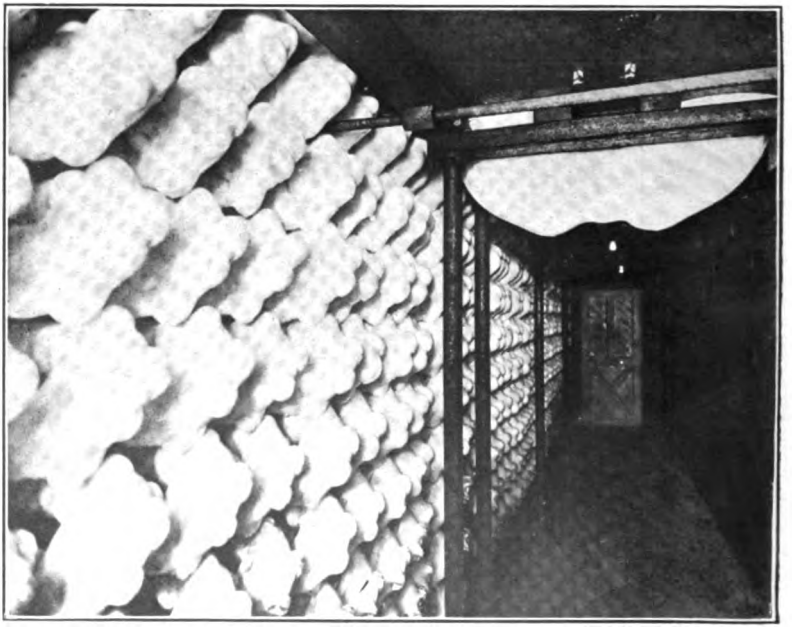
FIG. 8.



REVOLVING ELECTRIC FANS UNDER REFRIGERATOR COILS.

FIG. 9.



FROSTED ENDS OF COILS IN THE REFRIGERATING-CHAMBER.

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The plant is constructed throughout in the most substantial manner, for it is obvious that an apparatus treating such an important element of the process as the atmosphere could not be practically applied to a modern furnace in an experimental way, but must of necessity be as ample in capacity and as substantial in construction as any of the accessories of the furnace-stack now in use.

In order to show the arrangement of different parts of the plant, four photographic illustrations are given. Fig. 6 shows the ammonia-compressors; Fig. 7, a view of the top of the brine-cooling tank with its pipe-connections; Fig 8, the fans under the refrigerating-coils, which are used to circulate the air; and Fig. 9, the frosted ends of the coils in the refrigerating-chamber.

This dry-blast plant was put in operation August 11, 1904. The furnace was making a grade of iron suitable for the basic open hearth furnace, containing less than 1 per cent. of silicon, with an ore-mixture consisting of 50 per cent. of Mesabi ore, the balance being soft hematites from Michigan. The mixture showed a yield by analysis of 53.5 per cent. of iron. The coke used was shipped from two mines and varied considerably in ash. The quantity of ash present in these two coals averaged 10.5 and 12.5 per cent. respectively. In order to obtain correct data from the use of the dry-blast, it was determined beforehand that no changes in any particular were to be made in the operation of the furnace, other than the introduction of dry air, and this has been rigidly adhered to. In the data following, a comparison is made between the operations of the furnace using dry air after August 11th and those from August 1st to 11th, when the furnace was using the atmosphere under ordinary conditions. A comparison with the previous month would show a greater economy in coke, but since a change was made in the ore-mixture in the latter part of July —which gave a lower coke-consumption per ton of iron—a comparison of data when using dry air with that obtained in August prior to its use, and with the same ore-mixture, would more accurately show the benefits derived. The burden on the furnace from August 1st to 11th inclusive was as follows:

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Coke,	10,200 pounds.
Ore,	20,000 pounds.
Stone,	5,000 pounds.

On August 11th a 5-per-cent. increase in burden was put on the furnace and later in the day 33 per cent. of dry blast was used. As soon as this small quantity was introduced its effect was noticeable by a brightening of the tuyeres and an increasing temperature of the cinder. After this change in burden had come to work, the condition of the furnace being, if anything, still more satisfactory, an additional 5 per cent. of burden was put on, with confidence that an increased use of dry-blast would offset the increased duty on the furnace. From this period, on the burden and volume of dry-blast were increased more slowly until, on August 25th, the furnace, using dry-blast entirely, had the following burden:

Coke,	10,200 pounds.
Ore,	24,000 pounds.
Stone,	6,000 pounds.

an increase in burden of 20 per cent. in two weeks. The record of the furnace from August 1st to 11th, prior to the use of the dry-blast, and from August 25th to September 9th inclusive, using all dry-blast, is shown in Table VII.:

TABLE VII.—*Furnace Records, Without and With Dry Blast.*

WITHOUT DRY-BLAST.			WITH DRY-BLAST.		
Date.	Product.	Coke Consumption.	Date.	Product.	Coke Consumption.
	Tons.	Pounds.		Tons.	Pounds.
Aug. 1,	360	2,210	Aug. 25,	462	1,766
" 2,	367	2,112	" 26,	441	1,850
" 3,	372	2,084	" 27,	477	1,668
" 4,	373	2,133	" 28,	516	1,462
" 5,	386	2,008	" 29,	405	1,763
" 6,	340	2,280	" 30,	441	1,804
" 7,	347	2,116	" 31,	462	1,722
" 8,	360	2,012	Sept. 1,	472	1,729
" 9,	378	2,114	" 2,	472	1,642
" 10,	352	2,318	" 3,	458	1,648
" 11,	306	2,266	" 4,	421	1,841
			" 5,	450	1,813
Average,	358	2,147	" 6,	400	1,683
			" 7,	400	1,734
			" 8,	397	1,952
			" 9,	472	1,642
			Average,	447	1,726

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Fig. 10 shows graphically the record of furnace-operations from August 1st to September 9th inclusive; the increase in output and reduction in coke-consumption corresponding to the increase in burden; the varying conditions of humidity from day to day, which represent the average humidity for each twelve-hour period; and the change in humidity after treatment in the dry-blast apparatus. While the reduction of moisture and its increased uniformity is considerable, it should not be lost sight of that this represents the beginning of operations, and there was still much to be learned with respect to the manipulation of the dry-blast plant.

The effect of reducing and making more uniform the moisture in the blast, was clearly shown when, during a period of excessive humidity extending over three days, a neighboring furnace charged during this period an extra quantity of coke and increased the quantity each day, in order to maintain the grade of iron, while the Isabella furnace, operating with dry-blast, was in no wise affected.

On September 10th it was found necessary to make some repairs to the compressors and to make connections to a new brine-header for thawing-off the coils, and the burden was lightened accordingly. After these repairs had been made, the burden was again increased, and from September 17th to 30th inclusive the furnace showed an average daily output of 452 tons, with a coke-consumption of 1,729 lb. per ton of iron.

Of the changes made in the atmosphere by passing it through a refrigerating-chamber, the daily records of operations, set forth in Table VIII., will give a very clear idea.

During thirteen days, the average moisture in the atmosphere was 5.66 grains per cu. ft. and in the dry air 1.75 grains; 69 pounds of water were removed from the blast per ton of iron produced, which represents an average of 23,192 lb. (equivalent to 2,784 gals.) for the 24 hours. This weight was calculated from the volume of air blown into the furnace, as shown by piston-displacement. For four days during the above period the water caught in the tank underneath the refrigerating-chamber amounted to an average of 21,561 lb. (equivalent to 2,588 gals.) for the 24 hours, which is as close an agreement as could be expected, considering that the figures do not represent the same number of days, and that it is difficult to

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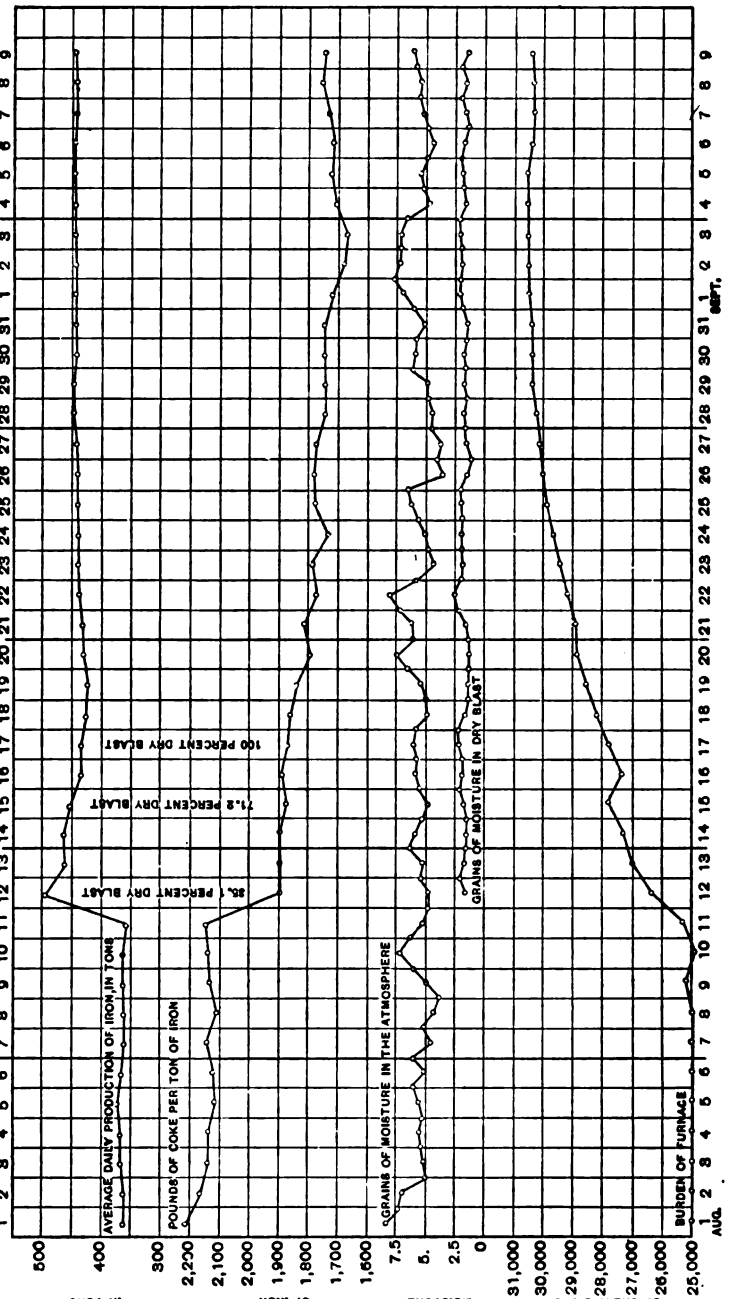

FIG. 10.

RECORD OF FURNACE-OPERATIONS, AUGUST 1 TO SEPTEMBER 9, 1904.

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TABLE VIII.—*Refrigerating-Chamber.*

Time.	Temperature.		Grains of Water per Cu. Ft. Air.		Temperature.		Grains of Water per Cu. Ft. Air.		Temperature.		Grains of Water per Cu. Ft. Air.	
	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.	Inlet.	Outlet.
6 A.M.	68	21	5.19	1.33	70	22	6.35	1.70	77	22	3.94	1.48
7 A.M.	68	20	5.02	1.24	71	22	6.78	1.77	.....	.....	4.08	1.29
8 A.M.	70	20	5.56	1.55	69	22	6.67	1.62	.....	.....	4.22	1.42
9 A.M.	73	20	5.37	1.46	73	22	6.78	1.70	71	25	4.85	1.36
10 A.M.	74	20	5.47	1.81	74	22	6.78	1.70	.....	.....	5.02	1.48
11 A.M.	77	20	5.56	1.53	77	23	6.67	1.70	.....	.....	5.19	1.55
12 M.	77	21	6.04	1.53	81	23	6.56	1.62	81	28	5.37	1.70
1 P.M.	80	21	6.04	1.42	78	24	6.56	1.70	.....	.....	4.85	1.62
2 P.M.	81	22	6.14	1.60	82	25	6.56	1.90	.....	.....	4.85	1.62
3 P.M.	81	23	5.74	1.60	81	24	6.19	1.74	84	29	5.02	1.70
4 P.M.	82	23	5.74	1.55	81	24	6.19	1.42	.....	.....	4.68	1.48
5 P.M.	82	22	6.04	1.62	80	24	6.14	1.48	.....	.....	4.85	1.60
6 P.M.	81	23	5.94	1.55	75	24	5.56	1.55	78	29	5.37	1.77
7 P.M.	80	23	5.74	1.62	72	24	5.94	1.70	.....	.....	5.37	1.62
8 P.M.	79	24	5.94	1.55	70	23	5.19	1.62	.....	.....	5.56	1.70
9 P.M.	73	23	7.01	1.85	69	22	5.19	1.42	72	29	5.74	1.70
10 P.M.	73	22	6.78	1.70	68	21	5.19	1.55	.....	.....	5.74	1.77
11 P.M.	73	23	6.78	1.70	66	20	3.94	1.77	.....	.....	5.74	1.62
12 NGT.	73	23	7.01	1.70	62	20	3.54	1.62	66	28	5.56	1.70
1 A.M.	73	23	6.78	1.70	59	18	3.41	1.42	.....	.....	4.85	1.70
2 A.M.	74	23	7.01	1.70	57	17	3.54	1.13	.....	.....	5.37	1.70
3 A.M.	73	23	6.78	1.70	56	16	3.18	1.13	64	27	5.19	1.48
4 A.M.	73	23	6.78	1.48	56	16	3.18	0.99	.....	.....	5.19	1.36
5 A.M.	73	23	6.78	1.48	53	14	2.85	1.06	.....	.....	4.85	1.48

determine accurately the volume and humidity of the air supplied in a given period. It is found sufficient in practice to thaw the frost off the pipes every three days. The coils are divided, for the purpose of thawing-off, into three sections, each representing the same number of coils, and a section is thawed each day; and in this way the work of refrigeration is not interfered with.

When the dry-blast was supplied to the furnace, it became necessary to reduce the revolutions of the blowing-engines, since the air supplied to the engines was lower in temperature than the natural atmosphere and contained more oxygen per cu. ft., and the tendency of the furnace was to drive too fast. Before applying the dry-blast the engines were running at 114 revolutions and supplying 40,000 cu. ft. of air per minute; the revolutions were gradually reduced to 96, thereby reducing the volume of blast over 6,000 cu. ft. per minute and increasing the effi-

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ciency of the engines by 14 per cent. With dried blast, 96 rev. per min. of the blowing-engines burned nearly 1 per cent. more coke and produced 89 tons more pig-iron in 24 hours than 114 rev. per min. with atmospheric air. The reduction in the revolutions resulted in a gain of 150 degrees in the temperature of the blast, which, even with this increase, through lack of area in the waste-gas ports of the stove, did not average above 870 degrees.

The average analysis of the gas for ten days prior to the introduction of the dry-blast showed:—CO, 22.3 per cent.; CO_2, 13 per cent., with an average temperature of 538 degrees. Later, with dry-blast used entirely, the average analysis was:—CO, 19.9 per cent; CO_2, 16 per cent., with an average temperature of 376 degrees. This reduction in temperature of 162 degrees is a necessary consequence of the greater concentration of heat in the hearth by the dry-blast combustion and the greater weight of burden heated by the gas, and represents an important saving of heat in the furnace.

The dry blast has resulted in economies in several other directions. In the use of Mesabi ore, which is very fine in structure, the waste of ore-dust carried by the escaping gases is quite large, and at many furnaces it has become quite burdensome. At the Isabella furnace, before dry-blast was used, it amounted to 5 per cent. of the ore charged; and it has been reduced, through the greater uniformity in the furnace-working, effected by the dry blast, to less than 1 per cent.

The saving in coke-consumption reduces the phosphorus in the metal; and this, in making Bessemer iron, permits the use of ores higher in phosphorus. As the Isabella furnace was making basic iron, it was an advantage to keep the silicon as low as possible, provided the sulphur was kept low; and the absence of irregularities in the furnace-operations resulting from the dry blast permitted the keeping of the silicon at a lower range without increasing the sulphur. It has been generally observed by furnace-managers that when the silicon is lowered through increased humidity in the atmosphere, a leaking tuyere, or through other causes, the sulphur is rapidly increased; but it has been found in using the dry blast, that when the hearth-temperature was suddenly lowered, principally from accretions on the bosh reaching the hearth, the sulphur did not increase;

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and in this respect the furnace has shown a remarkable uniformity in composition of the metal produced.

Mention has been made of the saving effected at the blowing-engines through a reduction in the number of the revolutions, which has an important bearing on the expenditure for power in operating the machines in the dry-blast plant. Prior to the use of the dry-blast plant, the average developed by each engine was 900 i.h.p. From the cards taken when the furnace was supplied with dry-blast, this average was 671 i.h.p., a difference of 229 i.h.p. per engine, or 687 i.h.p. for the three engines. Cards were also taken from the ammonia-compressors, the compression and back pressure being kept as nearly as possible to the best working-condition. Running at 45 revolutions, which would probably represent the average for the year, each engine developed 230 i.h.p., or 460 i.h.p. for the two engines; the fans, together with the brine and water-pumps, are well covered by allowing for them 75 i.h.p., making a total of 535 i.h.p. Comparing this with the power saved in the blowing-engine room, there appears a net saving, after allowing for the operation of the dry-blast plant. These figures, however, may not represent accurately the difference in power-consumption, as the blowing-engines were indicated at different times and the first test was taken with a blast-pressure on the furnace of 17 lb., while the test made with the dry blast was 15 lb., and the figures given above might require some modification, as the effect of dry blast on blast-pressure is not yet fully determined. The increase of uniformity in the working of the furnace, obtained through the dry blast, would result in a decrease of the blast-pressure; and I think in any event the saving in power-consumption in the blowing-engine room would nearly or quite compensate for the requirements of the dry-blast plant.

The application of the dry blast to the blast-furnace has shown, in addition to the economies effected, that the furnace can be controlled with precision; that it works with greater regularity; and that, in consequence, the product is uniform with respect to grade and composition, which makes the dry blast particularly valuable in the making of foundry iron, which is marketed by grade. An increase or decrease in blast tem-

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perature has a definite effect, and can be relied on to accomplish the desired result.

The dry-blast plant here described has been in regular operation since August 11, 1904. It started without a hitch and no difficulties have developed in any direction. Some modification in construction has been indicated as the result of the operation of the plant which would further reduce the moisture and add to its uniformity, but so far the changes suggested have been slight.

As above shown, the application of the dry blast to the blast-furnace has effected various economies and produced a more uniform metal. Probably its further application to the Bessemer converter would result in great benefit, since, in that apparatus, air is used in large quantities and the varying humidity affects the temperature of the charge and in consequence the quality of the steel. The metal from the metal-mixer employed in many Bessemer works is remarkably uniform; and the additional uniformity secured through the use of dry air would be of further advantage. In our American practice, a higher silicon is required in the summer months to maintain the temperature of the blow, in which period it is also more expensive to maintain the right amount of silicon in the pig-iron. With the use of the dry blast in the converter the proper temperature could be secured with a lower silicon in the metal, and this in turn would further reduce the coke-consumption at the furnaces. In other processes where air is used in large quantities—particularly in copper- and lead-smelters and copper-converters, in the open-hearth furnace and in cupolas—it seems probable that the use of dry-air would effect important economies, and its application to gas-producers, by-product coke- and charcoal-ovens, for the extraction of the moisture, would be very beneficial.

In the development of this process, which has occupied many years, during part of which time I was not directly in charge of blast-furnaces, much of the work has been necessarily entrusted to others. I desire to express my indebtedness to Mr. James Scott, General Superintendent of the Lucy and Isabella furnace-plants, and Mr. John P. Collins, Superintendent of the

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Isabella, for their valuable assistance in the application of the process; to Mr. Bruce Walter, Engineer at the works named, who conducted with great skill the later experiments, and supervised the construction of the dry-blast plant at the Isabella furnaces; and to Mr. John C. Greenway, then in the engineering department of the Duquesne steel-works, who conducted many of the earlier experiments.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Influence of Carbon, Phosphorus, Manganese and Sulphur on the Tensile Strength of Open-Hearth Steel.

BY H. H. CAMPBELL, STEELTON, PA.

(Lake Superior Meeting, September, 1904.)*

MANY attempts have been made to write a formula by which to calculate the strength of steel from its chemical composition, but most of these endeavors have failed because there were too many disturbing conditions. It is idle to collect from the pages of trade-papers, books, or the proceedings of scientific societies, a multitude of observations. The combination of such results will simply show that steel of the same composition will vary in tensile strength through wide limits—a fact that has been known for generations. That cold-working, overheating, and many another form of heat-treatment alter fundamentally the strength of steel, is ancient history, although it is only recently that the microscope has pointed out the road to an explanation of the phenomena.

It may be urged that the microscopic structure must be taken into consideration in any formula giving the ultimate strength, but from the standpoint of the present investigation this is unnecessary. We are trying to determine primarily, not what changes in ultimate strength may be made by variations in the condition of carbon, but what effect is produced by changes in the amount of carbon, when its condition remains constant. For such an inquiry, in order that the condition of the carbon should remain as nearly uniform as possible, it is essential that all test-pieces be made under the same conditions; and it is believed that the tests described in this paper satisfy that requirement. The investigations were made at the works of the Pennsylvania Steel Company, Steelton, Pa.; the ingots from which the tests were made were 6 in. square in every case; they were heated in the same furnace and forged at the same ham-

* Presented at the New York meeting of the Iron and Steel Institute (October, 1904), and here published under a mutual agreement between the Councils of the two Institutes.

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mer into billets of the same size; these billets were reheated in the same furnace by the same men and rolled in the same set of rolls into 2- by $\frac{3}{8}$-in. bars of about the same length. These were cooled under the same conditions, broken in the same machine by the same men, and analyzed in the same laboratory by the same staff.

First Investigation.

About ten years ago extensive calculations were made on such bars at Steelton by the method of least squares.[1] In this case the bars of similar composition were grouped together, and the carbon of the group was determined by combustion upon a sample containing an equal amount of drillings from each bar. Such a grouping of the tests would give rise to error if unlike bars were put together; so that this was avoided as far as possible. A greater cause of trouble lies in the fact that, in the method of least squares, any error in one factor affects the other factors. If for any reason the value of phosphorus comes out higher than the real truth, the value of carbon may suffer accordingly. Still greater is the objection that no factor can be used in the work unless it really has a decided influence on the results. This was illustrated when sulphur and copper were used as factors, the values found for them being absurd.

In acid steels, it appeared that carbon and phosphorus were the important elements to be considered; and it was assumed that the strength of steel was made up of the effect of a certain content of iron, plus the effect of a certain content of carbon, plus the effect of a certain content of phosphorus. Thus in a metal containing 0.20 per cent. of carbon and 0.08 per cent. of phosphorus, and having a tensile strength of 70,000 lb. per sq. in., we may write the following equation (the iron being determined by difference):

$20\ C + 8\ P + 9{,}972\ Fe = 70{,}000$; C, P and Fe being constants, representing the effect upon the tensile strength in pounds per square inch of 0.01 per cent. of carbon, phosphorus and iron respectively. In this way each group of tests furnished a formula and the combination of these by the method of least squares gave the following results:

[1] Full details of the work will be found in *The Manufacture and Properties of Iron and Steel*, pp. 482–528. Engineering & Mining Journal Company, New York, 1904.

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For Acid Steel: (C = 1,210; P = 890; Fe = 38,600.)

38,600 Iron + 1,210 Carbon + 890 Phosphorus + R = Ultimate Strength.

For Basic Steel: (C = 950; P = 1,050; Fe = 37,430, and Mn = 85.)

37,430 + 950 Carbon + 1,050 Phosphorus + 85 Manganese + R = Ultimate Strength.

The iron, carbon, phosphorus and manganese are expressed in units of 0.01 per cent., and R, as well as the ultimate strength, in pounds per square inch. R is a variable, to allow for heat treatment. In angles and plates, about $\frac{3}{8}$-in. or $\frac{1}{2}$-in. in thickness, and finished at a fairly high temperature, R is zero.

In this formula, Fe, 38,600, represents the value independently determined for pure iron. From a mathematical standpoint there can be no objection to including iron as one of the factors in the problem, but practically there are the following reasons to the contrary:

1. There is a doubt whether the real basis of strength varies with each increase or decrease in the total metallic iron. It may be that the datum-plane is the same, whether the steel contains 99.6 per cent. or 99.1 per cent. of iron.

2. Since the iron is determined by difference, all the errors in determining carbon, manganese and phosphorus, as well as the total contents of sulphur, copper, silicon, and other elements, may make a composite error of no small moment; and this is all embodied in the figure representing iron.

3. The range of variation in the percentage of iron is not sufficiently great to give a good working-basis.

Notwithstanding all these objections to the methods of determination, the derived formulæ given above have been of the utmost practical importance. They have been applied to every heat of steel made by the Pennsylvania Steel Company for the last ten years; and there has rarely been, in the ultimate strength of soft steels, a difference of more than 2,500 lb. per sq. in. between the result given by the formula and the record of the testing-machine. In most cases the error has been much less than 1,500 lb.; and so great is our confidence in the formulæ, that chemical determinations are always repeated when they are not confirmed by the machine-test. In view of such an experience in every-day commercial work, it would be

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rash to say that the method is entirely wrong, or that the formulæ do not represent actual conditions.

Second Investigation.

To check the first investigation, two entirely new series of bars were collected: one of nearly seven hundred from acid heats, and the other of eleven hundred from basic heats. Duplicate determinations were made on each bar for phosphorus and manganese. The carbon was determined in three ways: (1) the bar was analyzed by combustion (duplicate tests being made in case of doubt); (2) the bar was analyzed by the color-test; (3) a piece of the ingot from which the bar had been made was cut off at the hammer, and analyzed by the color-test.

Three bars were pulled on each heat, two on one testing-machine and one on another. The figure used is the average of the results obtained on the two machines, not the average of the three bars.

In order to plot the data, all heats were combined which showed carbon from 0.075 to 0.125 per cent.; from 0.125 to 0.175 per cent., and so on—making a division for each additional 0.05 per cent. of carbon. Table I gives the list of groups thus formed.

The division of the groups according to the way in which the carbon was determined is important, because many heats may change their grouping according to the way the carbon is determined; thus if a heat showed 0.12 per cent. of carbon by combustion, it would appear in Fig. 1 in line AA, at the point representing the range between 0.075 and 0.125 per cent. of carbon; while, if the color-determination showed 0.14 per cent., it would appear in the line BB at the point representing the range between 0.125 and 0.175 per cent. In this way the three sets of lines may be viewed as the result of three independent investigations.

The lines AA, BB and CC in Fig. 1 are not plotted directly from Table I, but the data have been combined by recognized scientific methods to allow for the unequal number of heats in the groups. Thus by combining groups 1, 2 and 3, we get the first point of AA; from groups 2, 3 and 4 we get the second point; and so on. The result of this combination gives Table II. The lines AA, BB, etc., plotted therefrom, are more nearly

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TABLE I.—*List of Groups Used in Determining the Effect of Carbon, Phosphorus and Manganese.*

Division.	Number of Heats.	Average Chemical Composition.			Ultimate Strength.
		Carbon.	Phosphorus.	Manganese.	
		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. In.
A, acid test-bars; carbon by combustion.	50	0.1118	0.0545	0.408	58,012
	131	0.1463	0.0567	0.437	61,089
	58	0.1995	0.0579	0.475	66,809
	22	0.2463	0.0563	0.484	70,736
	50	0.3065	0.0476	0.528	79,058
	120	0.3501	0.0466	0.537	83,093
	103	0.4000	0.0400	0.518	87,156
	86	0.4491	0.0376	0.520	92,824
	42	0.4961	0.0363	0.519	98,224
	8	0.5460	0.0354	0.495	102,346
	6	0.5863	0.0330	0.493	107,398
B, acid test-bars; carbon by color.	45	0.113	0.0545	.....	58,535
	164	0.145	0.0568	.....	62,407
	47	0.197	0.0560	.....	67,052
	8	0.249	0.0527	.....	72,728
	36	0.304	0.0494	.....	80,776
	53	0.352	0.0380	.....	86,369
	45	0.395	0.0358	.....	92,759
	18	0.444	0.0330	.....	98,576
	2	0.480	0.0360	.....	103,120
C, acid test-ingots; carbon by color.	34	0.118	.....	.....	57,599
	160	0.145	.....	.....	61,189
	61	0.190	.....	.....	67,482
	17	0.250	.....	.....	74,239
	84	0.307	.....	.....	80,491
	160	0.346	.....	.....	85,073
	98	0.397	.....	.....	91,434
	59	0.446	.....	.....	97,439
	15	0.507	.....	.....	105,656
D, basic test-bars; carbon by combustion.	135	0.0451	0.0082	0.243	46,703
	125	0.0974	0.0084	0.422	50,013
	134	0.1521	0.0116	0.436	55,650
	246	0.2044	0.0113	0.472	61,236
	263	0.2484	0.0110	0.474	64,744
	125	0.2935	0.0106	0.464	68,307
	27	0.3413	0.0113	0.461	72,065
	11	0.3932	0.0120	0.499	78,625
	1	0.4310	0.0070	0.390	83,305
E, basic test-bars; carbon by color.	173	0.047	0.0076	.....	47,084
	96	0.093	0.0100	.....	51,228
	189	0.154	0.0122	.....	58,202
	322	0.200	0.0118	.....	63,184
	235	0.248	0.0116	.....	65,813
	51	0.288	0.0125	.....	70,786
	3	0.343	0.0087	.....	79,252
F, basic test-ingots; carbon by color.	131	0.057	.....	.....	46,431
	131	0.093	.....	.....	49,917
	152	0.150	.....	.....	56,264
	365	0.203	.....	.....	62,241
	210	0.246	.....	.....	66,401
	72	0.295	.....	.....	71,011
	10	0.350	.....	.....	80,013
	2	0.400	.....	.....	80,272

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TABLE II.—*Combination of Data in Table I by Groups of Three, to Obtain Construction Points for the Lines in Fig.* 1.

Class.	Chemical Composition.			Ultimate Strength.
	Carbon.	Phosphorus.	Manganese.	
	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. In.
A, acid test-bars; carbon by combustion; line AA.	0.1520	0.0565	0.440	61,806
	0.1713	0.0570	0.453	63,637
	0.2486	0.0537	0.497	72,185
	0.3268	0.0480	0.529	80,626
	0.3609	0.0443	0.528	83,886
	0.3943	0.0419	0.526	87,155
	0.4357	0.0384	0.519	91,278
	0.4693	0.0371	0.518	95,052
	0.5130	0.0358	0.513	99,795
B, acid-test bars; carbon by color; line BB.	0.1489	0.0562	0.443	63,072
	0.1600	0.0564	0.453	64,379
	0.2437	0.0541	0.491	72,980
	0.3255	0.0434	0.519	83,168
	0.3534	0.0403	0.515	87,012
	0.3827	0.0364	0.513	90,742
	0.4112	0.0351	0.506	94,689
C, acid test-ingots; carbon by color; line CC.	0.152	.....	.....	62,216
	0.164	.....	.....	63,734
	0.257	.....	.....	74,937
	0.327	.....	.....	82,893
	0.351	.....	.....	85,771
	0.381	.....	.....	89,341
	0.423	.....	.....	94,734
D, basic test-bars; carbon by combustion; line DD.	0.0978	0.0094	0.366	50,834
	0.1639	0.0107	0.450	57,001
	0.2115	0.0113	0.465	61,502
	0.2403	0.0110	0.471	64,086
	0.2681	0.0109	0.470	66,297
	0.3081	0.0108	0.466	69,626
	0.3582	0.0113	0.469	74,203
E, basic test-bars; carbon by color; line EE.	0.1010	0.0101	0.384	52,540
	0.1688	0.0116	0.458	59,739
	0.2036	0.0118	0.466	62,750
	0.2260	0.0118	0.468	64,839
	0.2564	0.0117	0.469	66,830
F, basic test-ingots; carbon by color; line FF.	0.102	.....	.....	51,150
	0.168	.....	.....	58,339
	0.204	.....	.....	62,195
	0.227	.....	.....	64,575
	0.262	.....	.....	68,010
	0.304	.....	.....	72,303

representative of the true conditions, and are straighter than lines would be if plotted directly from Table I.

In Fig. 1 are shown all six lines. The line BB, founded on the color-determination of carbon in the acid test-bar, agrees very closely with CC, founded on color-carbons from the acid test-ingot. In the same way, but less closely, the line EE, representing color-carbons on the basic bar, agrees with the line

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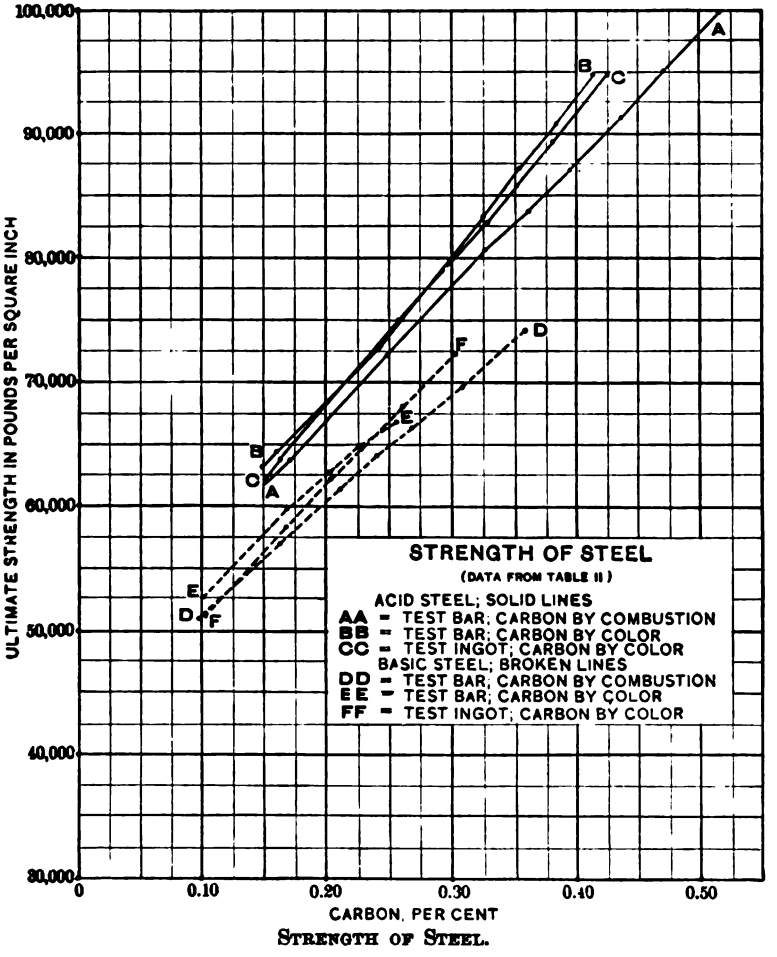
FF, representing color-carbons on the basic ingot. This additional investigation of the test-ingot was undertaken simply as a check on the other results; and, since it agrees as nearly as could be expected with the color-records on the bar, no further mention will be made of its results, comparison being made

FIG. 1.



STRENGTH OF STEEL.

only between the combustion- and the color-work on the test-bar. It may be well, however, to note that the line FF does not dip down at the upper end like EE, and this fact, taken in conjunction with the straightness of all the other lines, indicates that this dip in EE must be caused by errors in determination.

In the acid steels there is quite a difference between AA

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and BB, and in the basic steels between DD and EE. For a given amount of carbon, the results on the bar when analyzed by the color-method show a higher tensile strength than when carbon is determined by combustion, because a certain amount of temper-carbon is found by combustion which has little effect upon the ultimate strength. Determinations by color are often unsatisfactory; but when all the work is done in one laboratory, the results may be looked upon as comparable, and the reasonable straightness of the lines, as plotted, is good evidence that the chemical work was reliable.

The lines AA, BB, etc., take no account of variation in the content of phosphorus or of manganese. It is well known that phosphorus in small proportions adds to the tensile strength; in the first investigation by the method of least squares it was found that 0.01 per cent. raised the strength of acid steel 890 lb., and basic steel, 1,050 lb. per sq. in.; and experience has indicated that these values are very near the truth. In the present investigation the value of carbon is first determined, and then that of manganese and phosphorus, but in order to find the value of carbon accurately it is essential to know the influence of both manganese and phosphorus. This makes necessary the method of successive approximations, the values found in the first approximation being used in the second, and so on until the changes made in values are unimportant. In the present case the methods used avoid to some extent the dependence of one determination upon another. Thus in the line AA, carbon is the one great variable; the proportions of phosphorus and manganese are not constant, but the groups of high-carbon steel contain about the same amount of manganese and phosphorus as the groups of low-carbon steel, and hence the line will give a provisional value of carbon. The general trend is determined by stretching a thread along its length and noting the tangent made with the horizontal. In this way the line AA indicates a value for carbon of about 1,050 lb. for each 0.01 per cent.; allowances have yet to be made for the effect of phosphorus and manganese, but this figure serves as a working basis for similar provisional estimations of the other elements.

In explaining the method used to determine the value of phosphorus and manganese, no mention will be made of these

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provisional values, the figures given being in each case the final results.

The Effect of Phosphorus on Acid Steel.

The investigation into the effect of phosphorus will be confined to acid steel, for in the basic steels under consideration the proportions of phosphorus were so low that the differences were almost within the limits of chemical error. Two methods have been used, one serving as a check upon the other.

First Method.—Referring to Table I under the head of acid steels, carbon determined by combustion, the first group is composed of 50 heats averaging 0.1118 per cent. of carbon, 0.0545 per cent. of phosphorus, 0.408 per cent. of manganese and 58,012 lb. per sq. in. in ultimate strength. These 50 heats were separated into two groups, of high- and low-phosphorus respectively. The terms "high" and "low" are relative, and signify that the heats were arranged in a column according to their phosphorus-content, the upper half of the list being called "high" and the lower half "low." The result of this process applied to each group gave in each case two equal divisions showing certain differences in carbon, in manganese, in phosphorus and in ultimate strength. If the carbon and manganese had been uniform, the effect of phosphorus could have been found by simple division; but since they both varied, it was necessary to make allowance for them. If the difference in carbon were of any great amount a considerable error might be introduced; but as each group before division included only such heats as were within a range of 0.05 per cent. of carbon, the difference in this element in the high-phosphorus and low-phosphorus heats is in each case in the third decimal place.

Table III shows the first method of finding the value of phosphorus from the acid-steel bars in Table I. In each group we find the difference in ultimate strength, then allow for the variation in carbon, and in manganese, and find the difference in ultimate strength due to phosphorus alone; then divide by the variation in phosphorus and obtain the effect of one unit of phosphorus in the one group under consideration. If all the groups were composed of the same number of heats, it would suffice to take an average of these values of phosphorus, but as each group is made up of a different number, it is necessary to

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make a true average by multiplying the value found for each group by the number of heats in the group, and divide the sum of these results by the total number of heats. In this way the first method gives a value for phosphorus in acid steel of 807 lb. for 0.01 per cent.

TABLE III.—*Division of Heats to Determine the Effect of Phosphorus upon Acid Steel.*

NOTE.—In the sixth column an allowance is made of 1,000 lb. for 0.01 per cent. of carbon, and for manganese according to the sliding scale in Table VII.

Relative Phosphorus.	Chemical Composition.			Ultimate Strength.			Number of Heats in Group.	Product of Last Two Columns.
	Carbon.	Phosphorus.	Manganese.	Actual Records.	Difference Due to Phosphorus Alone.	Effect of 0.01 Per Cent. of Phosphorus.		
	Per Cent.	Per Cent.	Per Cent.	Lb. per Sq. Inch.	Lb. per Sq. Inch.	Lb. per Sq. Inch.		
High	0.1121	0.0635	0.405	58,543				
Low	0.1114	0.0456	0.411	57,481				
Diff	0.0007	0.0179	0.006	1,062	1,045	584	50	29,200
High	0.1470	0.0652	0.444	62,088				
Low	0.1457	0.0483	0.432	60,006				
Diff	0.0013	0.0169	0.012	2,082	1,808	1,070	131	140,170
High	0.1963	0.0667	0.481	67,443				
Low	0.2026	0.0490	0.470	66,174				
Diff	0.0063	0.0177	0.011	1,269	1,723	973	58	56,434
High	0.2459	0.0658	0.489	71,292				
Low	0.2467	0.0484	0.481	70,275				
Diff	0.0008	0.0174	0.008	1,017	937	539	22	11,858
High	0.3065	0.0564	0.546	80,111				
Low	0.3065	0.0399	0.509	78,005				
Diff		0.0165	0.037	2,106	1,218	738	50	36,900
High	0.3466	0.0545	0.540	83,669				
Low	0.3543	0.0372	0.534	82,410				
Diff	0.0077	0.0173	0.006	1,259	1,861	1,076	120	129,120
High	0.3969	0.0470	0.520	87,104				
Low	0.4031	0.0328	0.516	87,209				
Diff	0.0062	0.0142	0.004	−105	387	273	103	28,119
High	0.4511	0.0441	0.524	93,673				
Low	0.4473	0.0311	0.517	91,975				
Diff	0.0038	0.0130	0.007	1,698	1,066	820	86	70,520
High	0.4949	0.0407	0.521	98,318				
Low	0.4970	0.0327	0.517	98,145				
Diff	0.0021	0.0080	0.004	173	226	282	42	11,844
High	0.5430	0.0433	0.508	104,047				
Low	0.5508	0.0275	0.483	100,645				
Diff	0.0078	0.0158	0.025	3,402	3,082	1,951	8	15,608
High	0.5815	0.0415	0.500	109,367				
Low	0.5898	0.0288	0.490	106,412				
Diff	0.0083	0.0127	0.010	2,955	3,321	2,615	6	15,690
Total							676	545,463
Average								807

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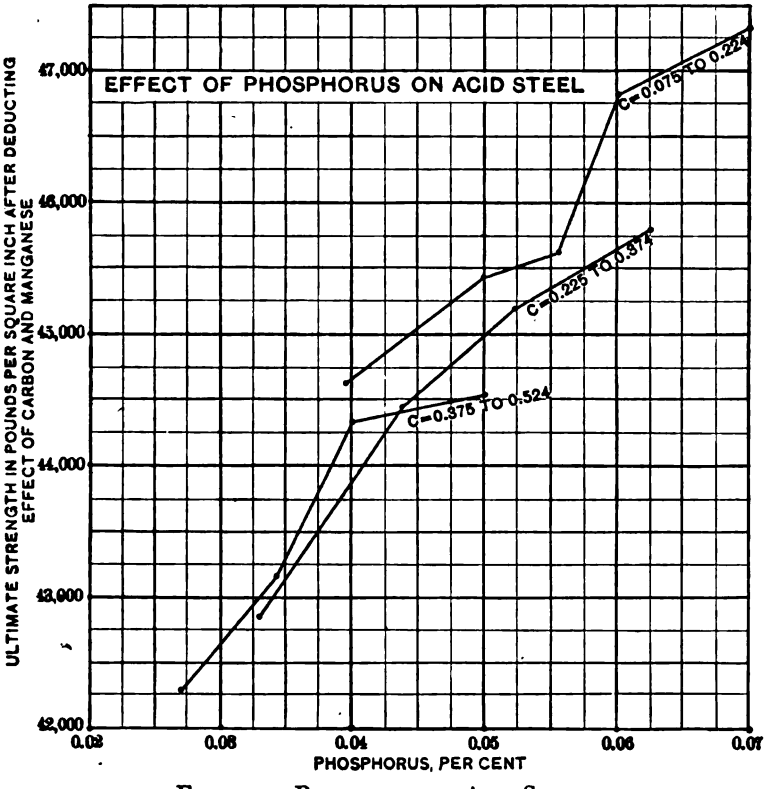
Second Method.—The bars were classified as before according to carbon, and then each of these main groups was subdivided according to phosphorus. Heats containing 0.03 per cent. of phosphorus constituted one group; those with 0.031 per cent. another; those with 0.032 per cent. another, and so on. Having made a list of these groups they were put together so as to give four or five points with about an equal number of heats in

FIG. 2.



EFFECT OF PHOSPHORUS ON ACID STEEL.

each, the result being shown in Table IV. In the last column is given what may be called the base, or the strength of the iron and phosphorus after allowing for carbon and manganese; this last column is plotted in Fig. 2. By combining the groups so as to rectify the lines by the method used in Table II, it will be found that in the line representing heats ranging between 0.075 and 0.224 per cent. of carbon, the phosphorus has a

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value of about 860 lb. for each 0.01 per cent.; in the range from 0.225 to 0.374 per cent. of carbon, the value is 940 lb.; between 0.375 and 0.524 per cent. of carbon it is 1,290 pounds. This would indicate that as the percentage of carbon increases, the effect of each unit of phosphorus increases, but the difference is so unimportant and the margin of certainty so narrow that it will be better to make a true average of the three values. There were 239 heats giving a value of 860 lb., 192 heats giving 940 lb., and 231 heats giving 1,290 lb., so that the true average is 1,033 lb. For the sake of simplicity the value of 0.01 per cent. of phosphorus will be taken as 1,000 pounds.

In reducing to a zero-base, as in the last column of Table IV, there will be certain errors, since the values of carbon and manganese are not inerrant; but the original classification into groups of about the same carbon minimizes the disturbing effect. Thus in Table IV the first main division has five units; the highest carbon is 0.1540 per cent. and the lowest 0.1491 per cent., a variation of 0.0049 per cent. Carbon has been valued at 1,000 lb. for 0.01 per cent., and if perchance that value is in error by 50 lb. the results determined from that division of the table will be wrong by only 50 x 0.49 = 25 lb. The last column shows a strength of 47,328 lb. for one base and 44,616 lb. for the other, a difference of 2,712 lb., so that the assumed error of 50 lb. in the value of carbon produces an error of only 1 per cent. in the value of phosphorus in this particular division. This argument applies also to the determination of the other elements in both acid and basic steel.

Another important consideration applying equally to the work on phosphorus and on manganese is the concordance of results obtained from different divisions. A general average obtained by grouping any data into two primal divisions gives conclusions of very limited value, but in this paper the practice is followed of subdividing in order to compare results. Thus from three independent lines of Fig. 2 the values of phosphorus varied from 860 to 1,290. It is quite possible that these variations were not accidental and that the variation represents a law of increasing effect with higher carbons, but leaving all this aside, it is certain that three separate determinations roughly agreeing with one another establish with reasonable certainty the general fact that 0.01 per cent. of phos

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phorus has a strengthening effect of somewhere about 1,000 lb. The validity of the conclusions is immeasurably superior to one based on a general average.

TABLE IV.—*Classification of Acid Heats According to Content of Phosphorus.*

NOTE.—In the last column a value of 1,000 lb. is given to 0.01 per cent. of carbon; the figure for manganese is taken from Table VII. Fig. 3 is plotted from the last column, but the data are combined to rectify the lines.

Limits of Carbon.	Number of Heats.	Chemical Composition.				Ultimate Strength.	
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	After Deducting for Carbon and Manganese.
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
0.075 to 0.224	39	0.1491	0.0396	0.439	0.0539	59,994	44,616
	54	0.1524	0.0500	0.430	0.0559	61,038	45,438
	38	0.1504	0.0557	0.441	0.0568	61,595	46,063
	61	0.1528	0.0617	0.445	0.0588	62,633	46,813
	47	0.1540	0.0717	0.447	0.0623	63,292	47,328
0.225 to 0.374	46	0.3373	0.0331	0.514	0.0477	79,636	42,805
	53	0.3317	0.0438	0.537	0.0529	81,231	44,444
	44	0.3265	0.0523	0.527	0.0538	81,197	45,194
	49	0.3120	0.0626	0.537	0.0537	80,390	45,792
0.375 to 0.524	52	0.4413	0.0271	0.514	0.0437	90,413	42,270
	63	0.4424	0.0343	0.508	0.0461	91,180	43,138
	54	0.4366	0.0404	0.521	0.0494	92,215	44,320
	62	0.4235	0.0504	0.534	0.0526	91,370	44,517

EFFECT OF MANGANESE ON ACID STEEL.

First Method.—The heats were divided into "low" and "high" manganese in the same way and within the same carbon-limits as already described in the determination of phosphorus. The results as given in Table V show that the effect of one unit of manganese is greater as the carbon increases. The increase is not regular, but this is partly explained by the small number of heats in some of the groups. Combining the data so as to have three larger groups, and plotting the results, it was found that for each increase of 0.01 per cent. of carbon, the effect of 0.01 per cent. of manganese was 10 lb. more. That is to say, if 0.01 per cent. of manganese strengthens a steel of 0.2 per cent. carbon by 160 lb., it will strengthen a steel of 0.21 per cent. of carbon by 170 pounds. In the second method of determining manganese it is found that the increment is 8 lb., which agrees fairly well with this valuation.

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TABLE V.—*Division of Heats to Determine the Effect of Manganese upon Acid Steel.*

NOTE.—In the eighth column a value is given both to carbon and phosphorus of 1,000 lb. for 0.01 per cent.

Limits of Carbon.	Relative Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	Difference Due to Manganese Alone.	Effect of 0.01 Per Cent. of Manganese.
Per Cent.			Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
	High.	27	0.1156	0.0564	0.440	58,847	.....	.....
0.075 to 0.124	Low..	23	0.1073	0.0524	0.370	57,031	.....	.....
	Diff...		0.0083	0.0040	0.070	1,816	586	84
	High.	63	0.1517	0.0572	0.476	62,008	.....	.....
0.125 to 0.174	Low..	68	0.1413	0.0562	0.402	60,142	.....	.....
	Diff...		0.0104	0.0010	0.074	1,866	726	98
	High.	27	0.1974	0.0590	0.514	66,936	.....	.....
0.175 to 0.224	Low..	31	0.2012	0.0567	0.440	66,698	.....	.....
	Diff...		0.0038	0.0023	0.074	238	388	52
	High.	10	0.2413	0.0551	0.519	70,602	.....	.....
0.225 to 0.274	Low..	12	0.2505	0.0574	0.456	70,850	.....	.....
	Diff..		0.0092	0.0023	0.063	−248	902	143
	High.	26	0.3048	0.0524	0.568	79,926	.....	.....
0.275 to 0.324	Low..	24	0.3083	0.0425	0.486	78,118	.....	.....
	Diff...		0.0035	0.0099	0.082	1,808	1,168	142
	High.	59	0.3513	0.0476	0.582	84,670	.....	.....
0.325 to 0.374	Low..	61	0.3489	0.0458	0.493	81,569	.....	.....
	Diff...		0.0024	0.0018	0.089	3,101	2,661	299
	High.	53	0.3987	0.0405	0.556	88,554	.....	.....
0.375 to 0.424	Low..	50	0.4014	0.0394	0.478	85,675	.....	.....
	Diff...		0,0027	0.0011	0.078	2,879	3,039	390
	High.	45	0.4492	0.0383	0.560	94,174	.....	.....
0.425 to 0.474	Low..	41	0.4490	0.0368	0.476	91,465	.....	.....
	Diff...		0.0002	0.0015	0.084	2,709	2,539	302
	High.	22	0.4970	0.0374	0.557	100,097	.....	.....
0.475 to 0.524	Low..	20	0.4948	0.0352	0.477	96,163	.....	.....
	Diff...		0.0022	0.0022	0.080	3,934	3,494	437
	High.	4	0.5500	0.0383	0.518	104,248	.....	.....
0.525 to 0.574	Low..	4	0.5440	0.0325	0.473	100,445	.....	.....
	Diff...		0.0060	0.0058	0.045	3,803	2.623	583
	High.	3	0.5827	0.0377	0.533	109,945	.....	.....
0.575 to 0.624	Low..	3	0.5897	0.0283	0.453	104,850	.....	.....
	Diff...		0.0070	0.0094	0.080	5,095	4,855	607

Second Method.—The heats were divided according to their content of manganese in the same way as explained in the the second determination of phosphorus. The results as given in Table VI and in Fig. 3 show that when the manganese exceeds 0.4 per cent. each increase in that element raises the strength, while with a content below 0.4 per cent. the tensile strength increases as the manganese decreases. The number of observations of low-manganese acid steels is not sufficient to

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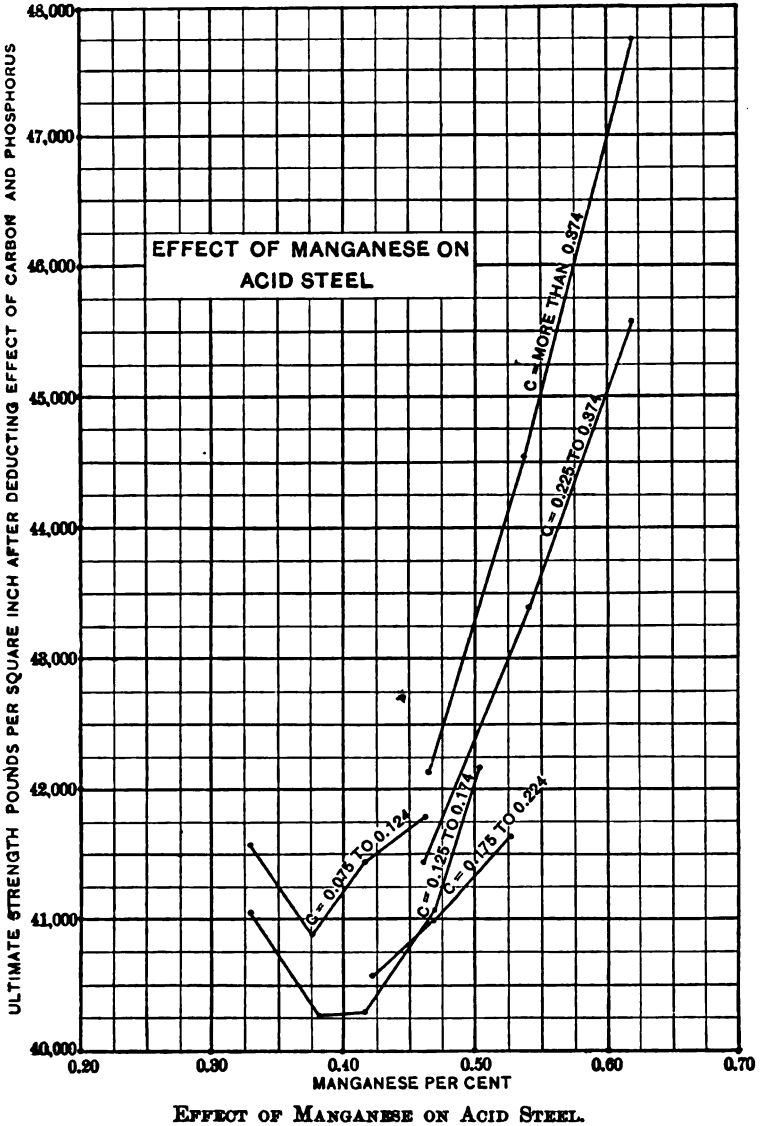
prove this conclusively, but on another page it will be shown that in basic steel also, a decrease in the manganese content below a certain point is not accompanied by a decrease in

FIG. 3.



EFFECT OF MANGANESE ON ACID STEEL.

strength. It is probable that low-manganese implies the presence of iron oxide and that this strengthens the steel much more than it is weakened by the decrease in manganese.

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The lines in Fig. 3 show that each increase in manganese above 0.4 per cent. is accompanied by an increase in strength, but this increase is not the same with steels of different carbon. In steels containing more than 0.374 per cent. of carbon, each increase of 0.01 per cent. of manganese augments the tensile strength by about 440 lb. per sq. in. In Table VI it is shown that the average carbon of this group is about 0.44 per cent. and we thus determine that for a steel of 0.44 per cent. of carbon, the strengthening effect of 0.01 per cent. of manganese is about 440 lb. per sq. in. In the same way the line of next lower carbon shows that in steels of 0.33 per cent. of carbon, the strengthening effect is about 260 lb. per sq. in. The next three lines may be considered as a unit indicating that for steels of 0.155 per cent. of carbon, the strengthening effect is about 125 lb. per sq. in. Plotting these data it was found that the strengthening effect of each 0.01 per cent. of manganese above a content of 0.4 per cent. is 80 lb. per sq. in. for a steel of 0.1 per cent. of carbon, but that for each rise of 0.01 per cent. of carbon the strengthening effect is increased 8 pounds. Thus an increase in manganese from 0.4 to 0.41 per cent. in steel of 0.1 per cent. of carbon raises the strength 80 lb., but an increase in manganese from 0.4 to 0.41 per cent. in steel of 0.11 per cent. of carbon raises the strength 88 pounds. A continuation of the line thus plotted gave zero-effect for zero carbon. With basic steel it will appear that a different value was obtained for a starting point and a different value for the increment. The law of variation in the effect of manganese upon acid steels is shown in Table VII.

Effect of Sulphur on Acid Steel.

First Method.—The heats were divided into high- and low-sulphur as shown in Table VIII following the same system as used with manganese and phosphorus. The results give a value of minus 100 lb. for 0.01 per cent. of sulphur, indicating that 0.01 per cent. weakens the steel by 100 lb. per sq. in.

Second Method.—The heats were classified according to their sulphur-content, the results being shown in Table IX and in Fig. 4. The second method corroborates the first in showing that sulphur has little influence upon the strength of acid steel.

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TABLE VI.—*Classification of Acid Heats According to Content of Manganese.*

NOTE.—In the last column both carbon and phosphorus are valued at 1,000 lb. for 0.01 per cent.

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.				Ultimate Strength.	
			Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	After Deducting for Carbon and Phosphorus.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
0.075 to 0.124	0.30 to 0.35	6	0.1052	0.0548	0.330	0.0560	57,558	41,558
	0.36 to 0.39	12	0.1117	0.0500	0.377	0.0576	57,047	40,877
	0.40 to 0.44	20	0.1110	0.0564	0.416	0.0589	58,173	41,433
	0.45 to 0.49	11	0.1168	0.0568	0.462	0.0636	59,135	41,775
0.125 to 0.174	0.30 to 0.35	2	0.1330	0.0585	0.330	0.0550	60,200	41,050
	0.36 to 0.39	19	0.1354	0.0538	0.381	0.0564	59,189	40,269
	0.40 to 0.44	55	0.1459	0.0569	0.417	0.0579	60,560	40,280
	0.45 to 0.49	41	0.1477	0.0564	0.470	0.0595	61,483	41,073
	0.50 to 0.59	14	0.1608	0.0601	0.503	.....	64,253	42,163
0.175 to 0.224	0.40 to 0.44	16	0.2004	0.0562	0.422	0.0504	66,237	40,577
	0.45 to 0.49	23	0.2016	0.0587	0.468	0.0567	67,020	40,990
	0.50 to 0.59	19	0.1960	0.0579	0.527	.....	67,035	41,645
0.225 to 0.374	0.40 to 0.49	47	0.3127	0.0476	0.461	.....	77,471	41,441
	0.50 to 0.59	122	0.3305	0.0482	0.541	.....	81,257	43,387
	0.60 to 0.69	19	0.3413	0.0476	0.618	.....	84,463	45,573
Over 0.374	0.40 to 0.49	83	0.4495	0.0359	0.465	.....	90,680	42,140
	0.50 to 0.59	144	0.4387	0.0395	0.537	.....	92,365	44,545
	0.60 to 0.69	17	0.4461	0.0387	0.618	.....	96,218	47,728

TABLE VII.—*Effect of Manganese upon Acid Steel.*

Carbon.	Manganese.										
	Per Cent. 0.40	Per Cent. 0.42	Per Cent. 0.44	Per Cent. 0.46	Per Cent. 0.48	Per Cent. 0.50	Per Cent. 0.52	Per Cent. 0.54	Per Cent. 0.56	Per Cent. 0.58	Per Cent. 0.60
Per Cent.		Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.
0.10	.....	160	320	480	640	800	960	1,120	1,280	1,440	1,600
0.15	.....	240	480	720	960	1,200	1,440	1,680	1,920	2,160	2,400
0.20	.....	320	640	960	1,280	1,600	1,920	2,240	2,560	2,880	3,200
0.25	.....	400	800	1,200	1,600	2,000	2,400	2,800	3,200	3,600	4,000
0.30	.....	480	960	1,440	1,920	2,400	2,880	3,360	3,840	4,320	4,800
0.35	.....	560	1,120	1,680	2,240	2,800	3,360	3,920	4,480	5,040	5,600
0.40	.....	640	1,280	1,920	2,560	3,200	3,840	4,480	5,120	5,760	6,400
0.45	.....	720	1,440	2,160	2,880	3,600	4,320	5,040	5,760	6,480	7,200
0.50	.....	800	1,600	2,400	3,200	4,000	4,800	5,600	6,400	7,200	8,000
0.55	.....	880	1,760	2,640	3,520	4,400	5,280	6,160	7,040	7,920	8,800
0.60	.....	960	1,920	2,880	3,840	4,800	5,760	6,720	7,680	8,640	9,600

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TABLE VIII.—*Division of Heats to Determine the Effect of Sulphur on Acid Steel.*

NOTE.—In the eighth column both carbon and phosphorus are valued at 1,000 lb. for 0.01 per cent.; the figure for manganese is taken from Table VII.

Limits of Carbon.	Relative Sulphur.	Chemical Composition.				Ultimate Strength.			Number of Heats.	Product of Last Two Columns.
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	Difference Due to Sulphur Alone.	Effect of 0.01 Per Cent. of Sulphur.		
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. In.	Pounds per Sq. In.	Pounds per Sq. In.		
	High.	0.1154	0.0587	0.424	0.0657	58,860	.....	.....	.....	.....
0.075 to 0.124	Low..	0.1079	0.0501	0.390	0.0523	57,093	.....	.....	.....	.....
	Diff...	**0.0075**	**0.0086**	**0.034**	**0.0134**	**1,767**	**−143**	**−107**	**50**	**−5,350**
	High.	0.1467	0.0600	0.439	0.0640	61,582	.....	.....	.....	.....
0.125 to 0.174	Low..	0.1485	0.0526	0.435	0.0509	60,398	.....	.....	.....	.....
	Diff...	**0.0018**	**0.0074**	**0.004**	**0.0131**	**1,184**	**+576**	**+440**	**131**	**+57,640**
	High.	0.2001	0.0604	0.488	0.0628	67,482	.....	.....	.....	.....
0.175 to 0.224	Low..	0.1987	0.0553	0.462	0.0490	66,181	.....	.....	.....	.....
	Diff...	**0.0014**	**0.0051**	**0.026**	**0.0138**	**1,301**	**+235**	**+170**	**58**	**+9,860**
	High.	0.2490	0.0590	0.456	0.0576	70,925	.....	.....	.....	.....
0.225 to 0.274	Low..	0.2435	0.0530	0.483	0.0488	70,550	.....	.....	.....	.....
	Diff...	**0.0055**	**0.0066**	**0.003**	**0.0088**	**375**	**−895**	**−1,017**	**22**	**−22,374**
	High.	0.3067	0.0526	0.538	0.0597	79,650	.....	.....	.....	.....
0.275 to 0.324	Low..	0.3062	0.0419	0.517	0.0475	78,363	.....	.....	.....	.....
	Diff...	**0.0005**	**0.0107**	**0.021**	**0.0122**	**1,287**	**−337**	**−276**	**50**	**−13,800**
	High.	0.3499	0.0496	0.546	0.0574	83,526	.....	.....	.....	.....
0.325 to 0.374	Low..	0.3503	0.0432	0.527	0.0460	82,615	.....	.....	.....	.....
	Diff...	**0.0004**	**0.0064**	**0.019**	**0.0114**	**911**	**−221**	**−194**	**120**	**−23,280**
	High.	0.3993	0.0437	0.527	0.0543	87,475	.....	.....	.....	.....
0.375 to 0.424	Low..	0.4007	0.0363	0.509	0.0424	86,830	.....	.....	.....	.....
	Diff...	**0.0014**	**0.0074**	**0.018**	**0.0119**	**645**	**−531**	**−446**	**103**	**−45,938**
	High.	0.4522	0.0418	0.529	0.0551	93,721	.....	.....	.....	.....
0.425 to 0.474	Low..	0.4462	0.0336	0.512	0.0413	92,082	.....	.....	.....	.....
	Diff...	**0.0060**	**0.0082**	**0.017**	**0.0138**	**1,639**	**−393**	**−285**	**86**	**−24,510**
	High.	0.4944	0.0380	0.518	0.0519	97,911	.....	.....	.....	.....
0.475 to 0.524	Low..	0.4975	0.0347	0.520	0.0426	98,536	.....	.....	.....	.....
	Diff...	**0.0031**	**0.0033**	**0.002**	**0.0093**	**−625**	**−565**	**−608**	**42**	**−25,536**
	High.	0.5421	0.0405	0.500	0.0510	103,665	.....	.....	.....	.....
0.525 to 0.574	Low..	0.5517	0.0302	0.490	0.0422	101,027	.....	.....	.....	.....
	Diff...	**0.0096**	**0.0103**	**0.010**	**0.0088**	**2,638**	**+2,128**	**+2,418**	**8**	**+19,344**
	High.	0.5897	0.0300	0.507	0.0503	108,550	.....	.....	.....	.....
0.575 to 0.624	Low..	0.5827	0.0360	0.480	0.0417	106,245	.....	.....	.....	.....
	Diff...	**0.0070**	**0.0060**	**0.027**	**0.0086**	**2,305**	**+952**	**+1,107**	**6**	**+6,642**
Total.....									676	−67,302
Average.....									.....	−100

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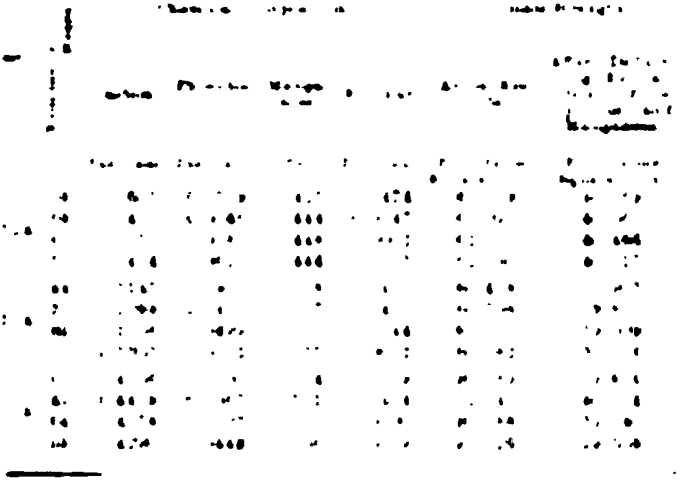
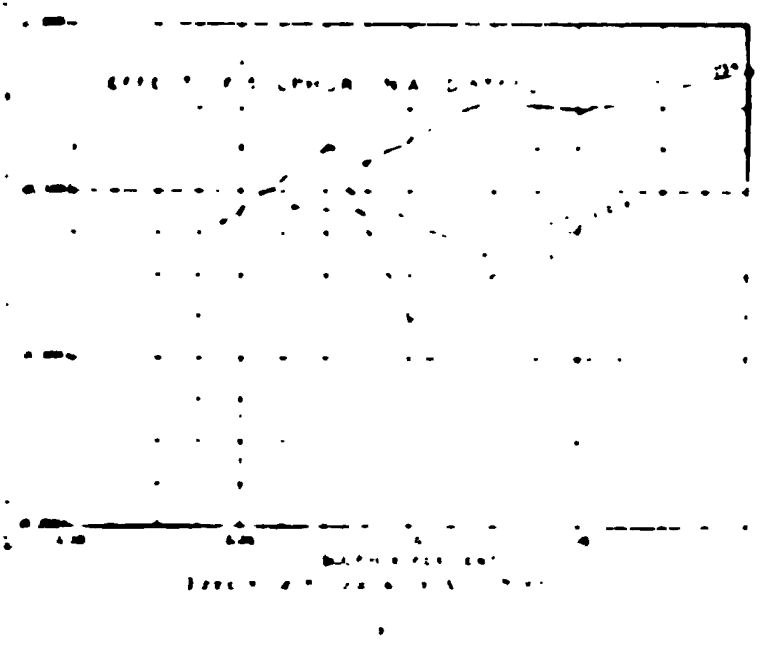
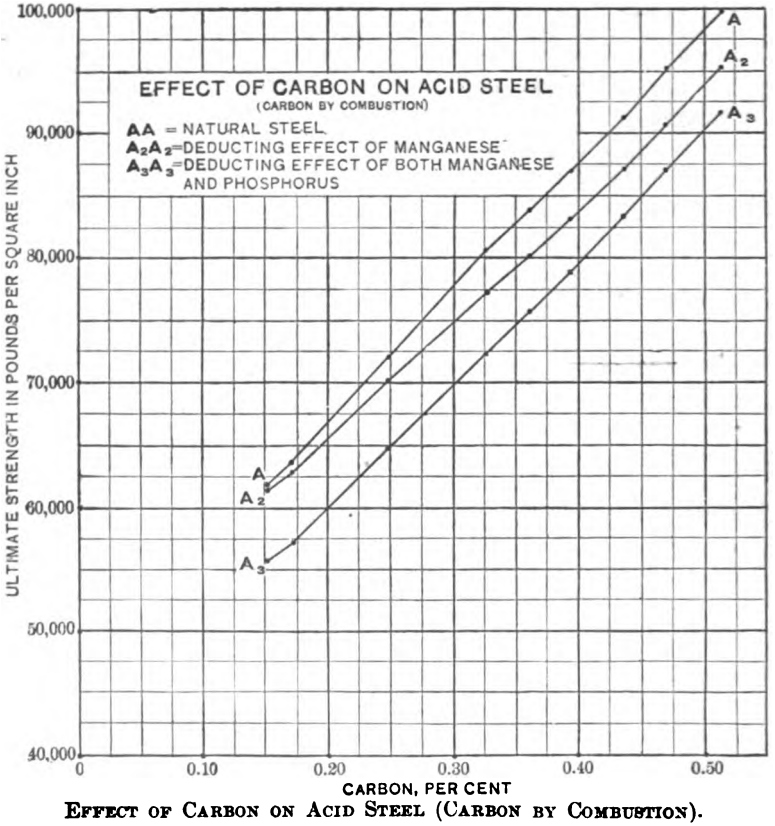




Effect of Carbon on Acid Steel.

Having found the effect of manganese and phosphorus, it becomes possible to correct the original line so as to determine the value of carbon. It so happens that the heats of higher carbon are at the same time of higher manganese. This makes a double correction, as an allowance must be made for the

Fig. 5.



Effect of Carbon on Acid Steel (Carbon by Combustion).

greater amount of manganese and for the greater effect of this element in steels of higher carbon. The allowances are made in accordance with Table VII. The result is to drop the upper end of the line more than the lower and thereby decrease the angle which the line makes with the horizontal, this angle measuring the effect of carbon.

In allowing for phosphorus very little change is made in

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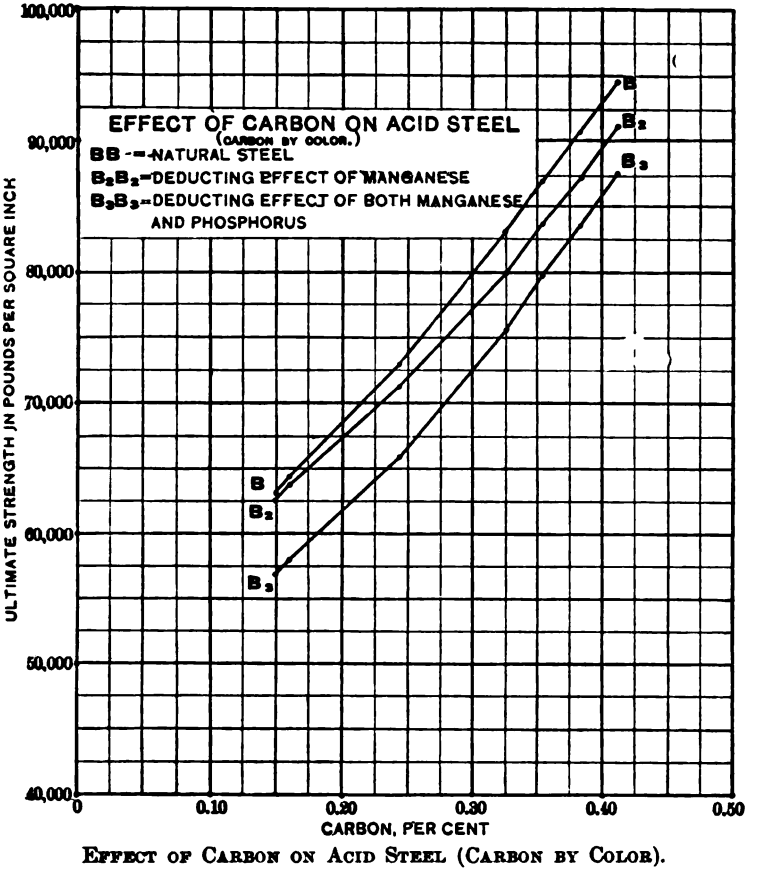
this angle as the proportion of phosphorus is nearly the same in the low- and in the high-carbon steels, but the whole line is lowered, thereby giving a lower value for the point where the prolongation of the line intersects the ordinate of zero carbon. Table X gives the corrected values which are plotted in Figs. 5 and 6. The lines AA and BB are copied from Fig. 1; the

FIG. 6.



EFFECT OF CARBON ON ACID STEEL (CARBON BY COLOR).

lines A_2A_2 and B_2B_2 represent the allowance for manganese, and the lines A_3A_3 and B_3B_3 are corrected for both manganese and phosphorus. The line A_3A_3 indicates a value of 1,000 lb. for each 0.01 per cent. of carbon, when the combustion method is used, and it intersects the zero ordinate at 40,000 lb. The line B_3B_3 indicates a value of 1,140 lb. for each 0.01 per cent. of

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carbon, when the color method is used, and it intersects the zero ordinate at 39,800 pounds.

TABLE X.—*Effect of Carbon upon Acid Steel.*

NOTE.—In calculating the last column a value of 1,000 lb. is given to 0.01 per cent. of phosphorus; manganese is rated according to Table VII.

Class.	Chemical Composition.			Ultimate Strength.	
	Carbon.	Phosphorus.	Manganese.	Actual Records.	After Deducting for Phosphorus and Manganese.
	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Acid test-bars; carbon by combustion.	0.1520	0.0565	0.440	61,806	55,676
	0.1713	0.0570	0.453	63,637	57,216
	0.2486	0.0537	0.497	72,185	64,875
	0.3268	0.0480	0.529	80,626	72,472
	0.3609	0.0443	0.528	83,886	75,744
	0.3943	0.0419	0.526	87,155	78,996
	0.4357	0.0384	0.519	91,278	83,273
	0.4693	0.0371	0.518	95,052	86,917
	0.5130	0.0358	0.513	99,795	91,605
Acid test-bars; carbon by color.	0.1489	0.0562	0.443	63,072	56,936
	0.1600	0.0564	0.453	64,379	58,061
	0.2437	0.0541	0.491	72,980	65,823
	0.3255	0.0434	0.519	83,168	75,686
	0.3534	0.0403	0.515	87,012	79,762
	0.3827	0.0364	0.513	90,742	83,667
	0.4112	0.0351	0.506	94,689	87,702

EFFECT OF MANGANESE ON BASIC STEEL.

First Method.—The bars were divided into high- and low-manganese as shown in Table XI. The figures show an increasing value for manganese as the carbon increases, thereby agreeing with the work on acid steel. In the first group, composed of heats of very soft steel, the value of manganese is practically zero. This is the same thing as saying that the decrease in manganese-content from 0.408 per cent. to 0.118 per cent. did not decrease the strength, which is entirely in accord with the theory before advanced that iron oxide strengthens steel.

Second Method.—The bars were classified according to their content of manganese as shown in Table XII and in Fig. 7. The line of very low-carbon and low-manganese steels shows that in the absence of manganese the strength is raised by iron oxide or by some other agent. In steels of higher carbon less

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TABLE XI.—*Division of Heats to Determine the Effect of Manganese upon Basic Steel.*

NOTE.—In the eighth column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus.

Limits of Carbon.	Relative Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	Difference Due to Manganese Alone.	Effect of 0.01 Per Cent. of Manganese.
Per Cent.			Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
	High.	58	0.0622	0.0094	0.408	48,152	.....	.....
0.020 to 0.074	Low..	77	0.0322	0.0073	0.118	45,669	.....	.....
			0.0300	0.0021	0.290	2,483	—37	—1
	High.	64	0.0968	0.0086	0.485	50,726	.....	.....
0.075 to 0.124	Low..	61	0.0980	0.0083	0.357	49,445	.....	.....
			0.0012	0.0003	0.128	1,281	1,177	92
	High.	68	0.1555	0.0121	0.497	56,680	.....	.....
0.125 to 0.174	Low..	66	0.1488	0.0112	0.374	54,887	.....	.....
			0.0067	0.0009	0.123	1,793	1,003	82
	High.	120	0.2049	0.0130	0.535	62,370	.....	.....
0.175 to 0.224	Low..	126	0.2040	0.0097	0.411	60,162	.....	.....
			0.0009	0.0033	0.124	2,208	1,598	129
	High.	128	0.2495	0.0123	0.532	67,058	.....	.....
0.225 to 0.274	Low..	135	0.2474	0.0098	0.415	62,528	.....	.....
			0.0021	0.0025	0.117	4,530	3,896	333
	High.	60	0.2907	0.0123	0.525	69,593	.....	.....
0.275 to 0.324	Low..	65	0.2960	0.0090	0.408	67,144	.....	.....
			0.0053	0.0033	0.117	2,449	2,286	195
	High.	14	0.3414	0.0099	0.516	73,116	.....	.....
0.325 to 0.374	Low..	13	0.3411	0.0126	0.401	70,932	.....	.....
			0.0003	0.0027	0.115	2,184	2,171	189
	High.	6	0.3938	0.0150	0.583	82,695	.....	.....
0.375 to 0.424	Low..	5	0.3924	0.0080	0.398	73,741	.....	.....
			0.0014	0.0070	0.185	8,954	7,691	416

oxygen is present owing to the protecting power of carbon, and the decrease in strength with decrease in manganese holds good down to a content of 0.3 per cent. Considering only the lines representing steels with from 0.075 to 0.224 per cent. and with from 0.225 to 0.374 per cent. of carbon, and pursuing the same course of reasoning as explained in the valuation of manganese in acid steels, it appears that above the limit of 0.3 per cent. of manganese, the effect of each unit of that element is greater in the steels of higher carbon. In the acid steel the value at zero carbon was zero, the effect of 0.01 per cent. of manganese in a steel of 0.1 per cent. of carbon was 80 lb., and this effect increased 8 lb. with each rise of 0.01 per cent. of carbon.

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TABLE XII.—*Classification of Basic Heats According to Content of Manganese.*

NOTE.—In the last column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus.

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.	
			Carbon.	Phosphorus.	Manganese.	Actual Records.	After Deducting for Carbon and Phosphorus.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Below 0.075	0.05 to 0.09	12	0.0297	0.0075	0.081	45,803	42,766
	0.10 to 0.14	56	0.0327	0.0073	0.120	45,674	42,426
	0.15 to 0.29	13	0.0388	0.0072	0.191	45,961	42,254
	0.30 to 0.39	16	0.0608	0.0097	0.354	48,034	42,390
	0.40 to 0.49	34	0.0632	0.0091	0.438	47,981	42,205
	0.50 to 0.59	4	0.0663	0.0133	0.508	51,133	44,698
0.075 to 0.224	0.20 to 0.29	7	0.1103	0.0079	0.259	50,056	40,773
	0.30 to 0.39	114	0.1458	0.0098	0.363	54,110	41,904
	0.40 to 0.49	242	0.1668	0.0099	0.441	57,036	43,203
	0.50 to 0.59	110	0.1744	0.0125	0.531	59,316	44,638
	0.60 to 0.69	26	0.1887	0.0154	0.622	61,862	45,793
0.225 to 0.374	0.30 to 0.39	61	0.2678	0.0089	0.365	63,858	42,349
	0.40 to 0.49	221	0.2689	0 0101	0.446	65,949	44,236
	0.50 to 0.59	102	0.2668	0.0130	0.532	67,565	45,723
	0.60 to 0.69	28	0.2695	0.0139	0.624	69,467	47,327

TABLE XIII.—*Effect of Manganese upon Basic Steel.*

Carbon.	Manganese.						
	Per Cent. 0.30	Per Cent. 0.35	Per Cent. 0.40	Per Cent. 0.45	Per Cent. 0.50	Per Cent. 0.55	Per Cent. 0.60
Per Cent.		Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
0.05	.....	550	1,100	1,650	2,200	2,750	3,300
0.10	.....	650	1,300	1,950	2,600	3,250	3,900
0.15	.....	750	1,500	2,250	3,000	3,750	4,500
0.20	.....	850	1,700	2,550	3,400	4,250	5,100
0.25	.....	950	1,900	2,850	3,800	4,750	5,700
0.30	.....	1,050	2,100	3,150	4,200	5,250	6,300
0.35	.....	1,150	2,300	3,450	4,600	5,750	6,900
0.40	.....	1,250	2,500	3,750	5,000	6,250	7,500

In basic steel the value of 0.01 per cent. of manganese at zero carbon is 90 lb; the effect per 0.01 per cent. of manganese

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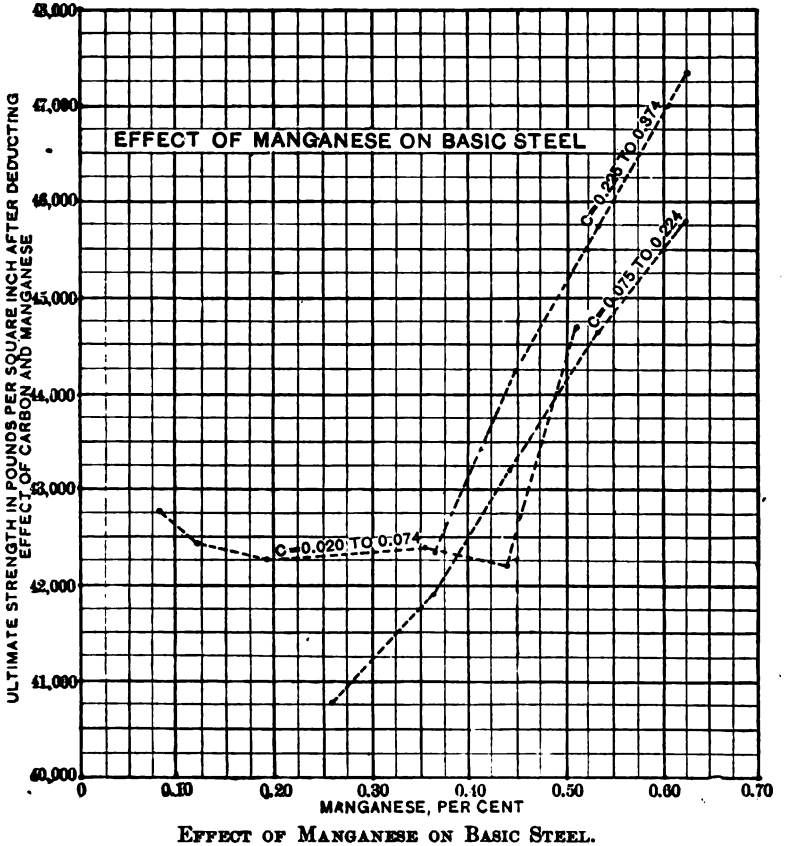
at 0.1 per cent. of carbon is 130 lb., and the increase in effect due to a rise of 0.01 per cent. of carbon is only 4 pounds. In the acid steel the base is 0.4 per cent. of manganese; in the basic steel it is 0.3 per cent. The results are tabulated in Table XIII.

FIG. 7.



EFFECT OF MANGANESE ON BASIC STEEL.

EFFECT OF SULPHUR UPON BASIC STEEL.

First Method.—The heats were divided into high- and low-sulphur, as shown in Table XIV, and the results indicate that 0.01 per cent. of sulphur strengthens steel by 58 lb. per sq. in. In the acid steel the same method of analysis showed a weakening effect of 100 lb. In either case the value is too small to be important.

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TABLE XIV.—*Division of Heats to Determine the Effect of Sulphur on Basic Steel.*

NOTE.—In the eighth column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus; manganese is rated as shown in Table XIII.

Limits of Carbon.	Relative Sulphur.	Chemical Composition.				Ultimate Strength.			Number of Heats.	Product of Last Two Columns.
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	Difference Due to Sulphur Alone.	Effect of 0.01 Per Cent. of Sulphur.		
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch,		
	High.	0.0545	0.0091	0.323	0.0353	47,512	.....	.....	.....	.....
0.020 to 0.074	Low..	0.0361	0.0074	0.166	0.0240	46,000	.....	*Note	.....	.....
	Diff...	0.0184	0.0017	0.157	0.0113	1,512	—74	—65	135	—8,775
	High.	0.1012	0.0088	0.417	0.0452	51,070	.....	.....	.....	.....
0.075 to 0.124	Low..	0.0936	0.0081	0.428	0.0267	49,116	.....	.....	.....	.....
	Diff...	0.0076	0.0007	0.011	0.0185	1,954	+1,442	+779	125	+97,375
	High.	0.1505	0.0133	0.428	0.0563	56,196	.....	.....	.....	.....
0.125 to 0.174	Low..	0.1540	0.0099	0.446	0.0342	55,394	.....	.....	.....	.....
	Diff...	0.0035	0.0034	0.018	0.0221	802	+1,002	+453	133	+60,250
	High.	0.2038	0.0124	0.479	0.0553	61,236	.....	.....	.....	.....
0.175 to 0.224	Low..	0.2050	0.0104	0.465	0.0356	61,247	.....	.....	.....	.....
	Diff...	0.0012	0.0020	0.014	0.0197	—11	—357	—181	246	—44,522
	High.	0.2475	0.0132	0.487	0.0517	65,016	.....	.....	.....	.....
0.225 to 0.274	Low..	0.2493	0.0089	0.457	0.0331	64,452	.....	.....	.....	.....
	Diff...	0.0018	0.0043	0.030	0.0186	564	—297	—160	263	—42,080
	High.	0.2922	0.0120	0.477	0.0447	68,771	.....	.....	.....	.....
0.275 to 0.324	Low..	0.2947	0.0092	0.451	0.0323	67,875	.....	.....	.....	.....
	Diff...	0.0025	0.0028	0.026	0.0124	896	+273	+220	125	+27,500
	High.	0.3386	0.0128	0.450	0.0414	71,525	.....	.....	.....	.....
0.325 to 0.374	Low..	0.3441	0.0093	0.472	0.0294	72,647	.....	.....	.....	.....
	Diff...	0.0055	0.0035	0.022	0.0120	—1,122	—551	—459	27	—12,392
	High.	0.3950	0.0145	0.497	0.0397	78,400	.....	.....	.....	.....
0.375 to 0.424	Low..	0.3910	0.0086	0.502	0.0308	78,895	.....	.....	.....	.....
	Diff...	0.0040	0.0059	0.005	0.0089	—495	—1,270	—1,427	11	—15,697
Total.....									1,065	+61,659
Average.....										+58

*NOTE.—In calculating the difference due to sulphur in the group of very low carbon steels, no allowance is made for the difference in manganese, as a decrease in manganese to a content of 0.166 per cent. is not necessarily followed by a decrease in strength.

Second Method.—The heats were classified according to their sulphur-content, as shown in Table XV and in Fig. 8. The lines are irregular and indeterminate, indicating a very small value for this element.

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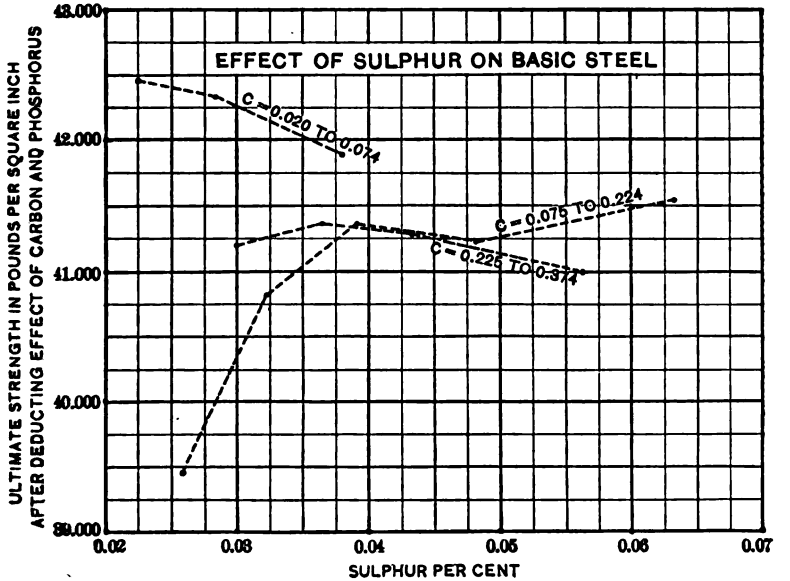
TABLE XV.—*Classification of Basic Heats According to Content of Sulphur.*

NOTE.—In the last column a value of 770 lb. is given to 0.01 per cent. of carbon, and 1,000 lb. to 0.01 per cent. of phosphorus; manganese is rated as shown in Table XIII.

Limits of Carbon.	Number of Heats.	Chemical Composition.				Ultimate Strength.	
		Carbon.	Phosphorus.	Manganese.	Sulphur.	Actual Records.	After Deducting for Carbon, Phosphorus and Manganese.
Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Below 0.075	45	0.0361	0.0074	0.162	0.0225	45,978	42,458
	46	0.0418	0.0077	0.212	0.0283	46,337	42,348
	44	0.0575	0.0096	0.356	0.0380	47,922	41,896
0.075 to 0.224	74	0.1225	0.0078	0.434	0.0258	51,524	39,462
	103	0.1571	0.0089	0.444	0.0322	56,027	40,822
	112	0.1786	0.0114	0.466	0.0391	58,944	41,363
	105	0.1790	0.0115	0.461	0.0482	58,767	41,226
	110	0.1696	0.0129	0.441	0.0632	58,129	41,552
0.225 to 0.374	115	0.2754	0.0083	0.453	0.0298	66,333	41,206
	113	0.2693	0.0097	0.458	0.0365	66,194	41,360
	89	0.2679	0.0114	0.464	0.0434	66,307	41,292
	98	0.2582	0.0149	0.504	0.0563	66,334	41,005

FIG. 8.



EFFECT OF SULPHUR ON BASIC STEEL.

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Effect of Carbon upon Basic Steel.

The effect of carbon was found, as in the case of acid steels, by allowing for phosphorus and manganese in the groups given in Table II. The data are given in Table XVI and in Figs. 9 and 10. The line D_3D_3 indicates a value of 770 lb. for each 0.01 per cent. of carbon when the combustion method is used and it intersects the zero ordinate at 41,500 pounds.

The line E_3E_3 indicates a value of 820 lb. for each 0.01 per cent. of carbon when the color-method is used, and it intersects the zero-ordinate at 42,000 pounds.

Table XVI.—*Effect of Carbon upon Basic Steel.*

Note.—In calculating the last column a value of 1,000 lb. is given to 0.01 per cent. of phosphorus; the manganese is rated as shown in Table XIII.

Class.	Chemical Composition.			Ultimate Strength.	
	Carbon.	Phosphorus.	Manganese.	Actual Records.	After Deducting for Phosphorus and Manganese.
	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Basic test-bars; carbon by combustion.	0.0978	0.0094	0.366	50,834	49,036
	0.1639	0.0107	0.450	57,001	53,621
	0.2115	0.0113	0.465	61,502	57,501
	0.2403	0.0110	0.471	64,086	59,805
	0.2681	0.0109	0.470	66,297	61,841
	0.3081	0.0108	0.466	69,626	64,994
	0.3582	0.0113	0.469	74,203	69,118
Basic-test bars; carbon by color.	0.1010	0.0101	0.384	52,540	50,438
	0.1688	0.0116	0.458	59,739	56,083
	0.2036	0.0118	0.466	62,750	58,748
	0.2260	0.0118	0.468	64,839	60,601
	0.2564	0.0117	0.469	66,830	62,381

It has already been explained, and is shown by Fig. 9, that any changes in the value of manganese affects the tangent of the carbon-line, thereby affecting the value found for a unit of that element; and as manganese has been given a slightly higher value in basic than in acid steel, it would naturally follow that the result for carbon would be lower in the basic than in the acid steel. To find how much this change in the value of manganese affected the carbon determination, the experiment was tried of correcting the line of basic, according to the values of manganese found for acid steel. The result showed a value of 810 lb. for 0.01 per cent. of carbon instead of 770 lb., as found

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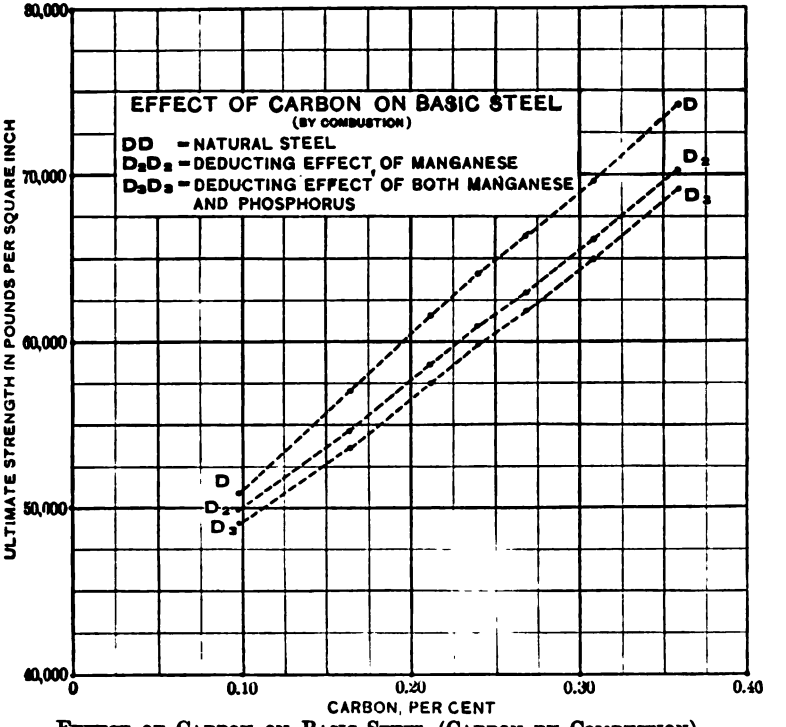
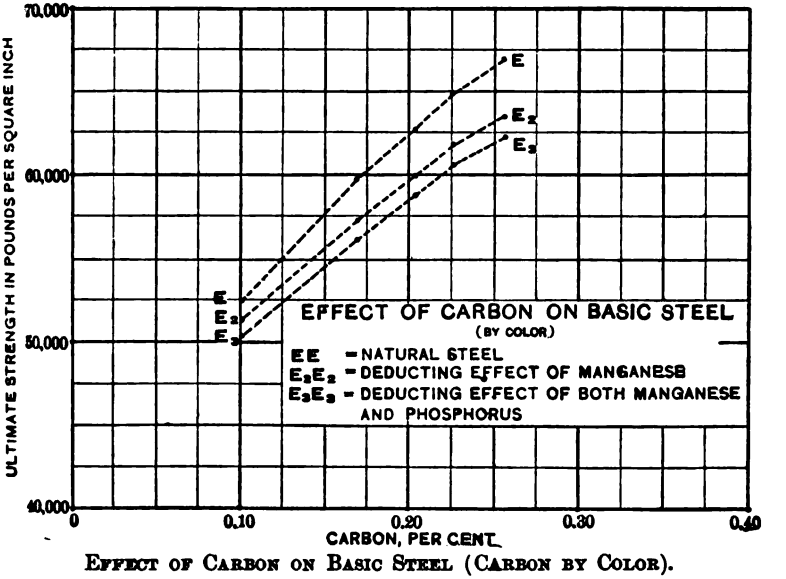
FIG. 9.



EFFECT OF CARBON ON BASIC STEEL (CARBON BY COMBUSTION).

FIG. 10.



EFFECT OF CARBON ON BASIC STEEL (CARBON BY COLOR).

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by the above special investigation. Inasmuch as the acid steel gave a value for carbon of 1,000 lb. per unit of 0.01 per cent., and as the basic steel gives 810 lb. when calculated by the acid formula and 770 lb. by its own formula, it would seem certain that a unit of carbon has much less effect upon basic than upon acid steel.

The Application of the Formulæ.

Table XVII shows the result of comparing the actual strength of the steels under consideration with the strength as calculated from the formulæ just given. For this purpose the heats were grouped according to carbon and then subdivided according to manganese. No heats were put together that varied more than 0.05 per cent. in carbon, or more than 0.1 per cent. in manganese. For instance, a group might include a heat containing 0.1 per cent. of carbon and 0.3 per cent. of manganese, and another heat containing 0.149 per cent. of carbon and 0.399 per cent. of manganese, but any heat of higher or lower carbon, or of higher or lower manganese than these extremes, would fall into another group. Inasmuch as the phosphorus did not vary through wide limits in any of the steels, each group may be looked upon as composed of heats that are practically alike in chemical composition, and which may properly be averaged to eliminate accidental errors.

In some of the subdivisions the number of heats is so small that these errors cloud the result. Especially in the steels of higher carbon it is desirable to have a large number of heats in the average, as it is difficult to get uniform results on a testing-machine under usual working conditions when the bar has a strength of over 90,000 lb. per sq. in., and unfortunately it is in these high steels and particularly in the groups with an unusual content of manganese that only a small number of heats were on record. There are accordingly several instances where these small groups show a considerable difference between the actual and the calculated strength, but there seems to be no rule as to the difference, as other groups, either large or small, of the same class of steels give satisfactory results.

It is of course a matter of opinion as to what constitutes a fair agreement between the actual and the calculated strengths, but in the following comparison it will be assumed that the

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TABLE XVII.—*Comparison of the Actual Ultimate Strength of Certain Groups of Steel with the Strength as Calculated from the Following Formulæ:*

Acid steel: 40,000 + 1,000 C + 1,000 P + x Mn = ultimate strength.
Basic steel: 41,500 + 770 C + 1,000 P + y Mn = ultimate strength.

Value of x as per Table VII; value of y as per Table XIII.

Heavy type denote that the difference between the actual and calculated strengths is over 1,500 pounds.

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	By Formula.	Difference.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Acid steel:								
0.075 to 0.124	0.30 to 0.39	18	0.1095	0.0517	0.361	57,217	56,120	—1,097
	0.40 to 0.49	31	0.1131	0.0566	0.432	58,414	57,258	—1,156
	0.50 to 0.59	1	0.1130	0.0440	0.500	56,745	56,600	—145
0.125 to 0.174	0.30 to 0.39	21	0.1352	0.0542	0.377	59,285	58,940	—345
	0.40 to 0.49	96	0.1466	0.0567	0.440	60,954	60,794	—160
	0.50 to 0.59	14	0.1608	0.0601	0.513	64,253	63,536	—717
0.175 to 0.224	0.40 to 0.49	39	0.2011	0.0577	0.449	66,698	66,664	—34
	0.50 to 0.59	19	0.1960	0.0579	0.527	67,035	67,371	+336
0.225 to 0.274	0.30 to 0.39	1	0.2340	0.0550	0.390	68,460	68,900	+440
	0.40 to 0.49	11	0.2520	0.0576	0.462	71,068	72,200	+1,132
	0.50 to 0.59	10	0.2413	0.0551	0.519	70,602	71,925	+1,323
0.275 to 0.324	0.40 to 0.49	14	0.3093	0.0446	0.469	78,200	77,101	—1,099
	0.50 to 0.59	32	0.3066	0.0485	0.541	79,167	78,950	—217
	0.60 to 0.69	**3**	**0.2863**	**0.0497**	**0.613**	**80,223**	**78,456**	**—1,767**
	0.70 to 0.79	**1**	**0.3240**	**0.0560**	**0.720**	**84,100**	**86,320**	**2,220**
0.325 to 0.374	**0.30 to 0.39**	**1**	**0.3490**	**0.0340**	**0.300**	**81,650**	**78,300**	**—3,350**
	0.40 to 0.49	22	0.3452	0.0446	0.455	80,208	80,498	+290
	0.50 to 0.59	80	0.3512	0.0472	0.544	83,425	83,872	+447
	0.60 to 0.69	16	0.3516	0.0472	0.619	85,258	86,012	+754
	0.70 to 0.79	1	0.3440	0.0450	0.700	86,840	87,180	+340
0.375 to 0.424	0.40 to 0.49	34	0.4009	0.0377	0.464	85,205	85,908	+703
	0.50 to 0.59	63	0.3996	0.0410	0.537	87,880	88,444	+564
	0.60 to 0.69	6	0.3993	0.0425	0.622	90,598	91,284	+686
0.425 to 0.474	0.40 to 0.49	27	0.4481	0.0363	0.462	90,950	90,672	—278
	0.50 to 0.59	53	0.4515	0.0382	0.539	93,760	93,974	+214
	0.60 to 0.69	6	0.4332	0.0378	0.617	93,805	94,587	+782
0.475 to 0.524	0.30 to 0.39	1	0.4770	0.0330	0.380	90,775	91,000	+225
	0.40 to 0.49	12	0.4955	0.0340	0.468	95,745	95,643	—102
	0.50 to 0.59	25	0.4961	0.0376	0.533	98,699	98,637	—62
	0.60 to 0.69	**4**	**0.5010**	**0.0365**	**0.617**	**104,550**	**102,430**	**—2,120**
0.525 to 0.574	0.40 to 0.49	6	0.5463	0.0303	0.478	100,718	101,061	+343
	0.50 to 0.59	2	0.5490	0.0505	0.545	107,230	106,330	—900
0.575 to 0.624	0.40 to 0.49	4	0.5887	0.0312	0.462	105,131	104,904	—227
	0.50 to 0.59	**1**	**0.5770**	**0.0430**	**0.510**	**112,760**	**107,071**	**—5,689**
	0.60 to 0.69	1	0.5850	0.0300	0.600	111,100	110,860	—240

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Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	By Formula.	Difference.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Basic steel:								
0.020 to 0.074	0.00 to 0.09	12	0.0297	0.0075	0.081	45,803	44,537	—1,266
	0.10 to 0.19	65	0.0326	0.0073	0.125	45,645	44,740	—905
	0.20 to 0.29	4	0.0543	0.0073	0.263	47,094	46,411	—683
	0.30 to 0.39	16	0.0608	0.0097	0.354	48,034	47,767	—267
	0.40 to 0.49	34	0.0632	0.0091	0.438	47,981	48,849	+868
	0.50 to 0.59	4	0.0663	0.0133	0.508	51,133	50,389	—744
0.075 to 0.124	0.10 to 0.19	1	0.0990	0.0080	0.160	45,780	49,923	+4,143
	0.20 to 0.29	6	0.0993	0.0078	0.262	49,378	49,926	+548
	0.30 to 0.39	42	0.0983	0.0086	0.363	49,683	50,748	+1,065
	0.40 to 0.49	53	0.0955	0.0083	0.438	49,667	51,477	+1,810
	0.50 to 0.59	21	0.0998	0.0089	0.539	51,900	53,182	+1,282
	0.60 to 0.69	2	0.0950	0.0085	0.660	55,773	54,345	—1,428
0.125 to 0.174	0.10 to 0.19	1	0.1370	0.0070	0.160	52,295	52,749	+454
	0.30 to 0.39	41	0.1486	0.0107	0.359	54,738	54,897	+159
	0.40 to 0.49	64	0.1531	0.0114	0.445	55,800	56,596	+796
	0.50 to 0.59	24	0.1549	0.0130	0.535	57,050	58,300	+1,250
	0.60 to 0.69	3	0.1657	0.0213	0.640	59,943	61,693	+1,750
0.175 to 0.224	0.20 to 0.29	1	0.1760	0.0080	0.240	54,120	55,852	+1,732
	0.30 to 0.39	31	0.2064	0.0104	0.367	59,276	59,611	+335
	0.40 to 0.49	125	0.2040	0.0098	0.441	60,752	60,670	—82
	0.50 to 0.59	65	0.2059	0.0135	0.527	62,547	62,698	+151
	0.60 to 0.69	21	0.2009	0.0152	0.616	62,716	63,987	+1,271
	0.70 to 0.79	3	0.2050	0.0087	0.713	65,507	65,424	—83
0.225 to 0.274	0.20 to 0.29	1	0.2300	0.0070	0.260	61,090	59,909	—1,181
	0.30 to 0.39	39	0.2458	0.0079	0.365	62,185	62,463	+278
	0.40 to 0.49	137	0.2489	0.0105	0.451	64,425	64,644	+219
	0.50 to 0.59	66	0.2490	0.0132	0.529	66,107	66,436	+329
	0.60 to 0.69	18	0.2495	0.0141	0.627	67,048	68,465	+1,417
	0.70 to 0.79	1	0.2740	0.0140	0.710	74,970	72,362	—2,608
	0.90 to 0.99	1	0.2280	0.0150	0.940	67,595	72,395	+4,800
0.275 to 0.324	0.30 to 0.39	18	0.2986	0.0085	0.366	65,920	66,753	+833
	0.40 to 0.49	70	0.2937	0.0098	0.440	67,888	68,063	+175
	0.50 to 0.59	29	0.2907	0.0128	0.540	69,725	70,202	+477
	0.60 to 0.69	8	0.2900	0.0142	0.621	72,402	71,991	—411
0.325 to 0.374	0.30 to 0.39	4	0.3443	0.0200	0.355	70,954	71,286	+332
	0.40 to 0.49	14	0.3396	0.0086	0.437	71,170	71,660	+490
	0.50 to 0.59	7	0.3354	0.0114	0.524	72,365	73,572	+1,207
	0.60 to 0.69	2	0.3675	0.0105	0.610	79,515	78,286	—1,229
0.375 to 0.424	0.30 to 0.39	2	0.3830	0.0080	0.355	73,620	73,154	—466
	0.40 to 0.49	5	0.3936	0.0102	0.448	75,107	76,555	+1,448
	0.50 to 0.59	1	0.3860	0.0110	0.500	79,750	77,280	—2,470
	0.60 to 0.69	2	0.4065	0.0220	0.645	88,545	83,832	—4,713
	0.70 to 0.79	1	0.3920	0.0080	0.750	85,260	83,732	—1,528

results of the formulæ should be within 1,500 lb. of the records of the testing-machine. In the acid steels there are 12 groups containing less than 5 heats each. In 7 of these the cal-

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culated strength agrees with the actual strength within 1,500 pounds. In 5 groups the difference is over 1,500 pounds. In the basic steel there are 17 groups containing less than 5 heats and 9 of these agree within 1,500 pounds. Eight groups show a difference greater than this amount. Taking both acid and basic steels, out of 29 "small" groups 16 are correct, and of the 13 that are beyond the limit 9 are single heats, most of them being steel of moderately high carbon.

In the acid steel there are 23 groups containing over 4 heats each, and all of them are within the limit of 1,500 lb., only 5 having an error exceeding 1,000 pounds. In the basic steel there are 26 groups with over 4 heats and 25 are within 1,500 lb., and 17 within 1,000 pounds. There is 1 group of 53 heats, averaging about 0.1 per cent. of carbon, which shows an error of + 1,810 pounds. Putting aside mathematical errors which can hardly be present in this investigation (owing to repeated checking of the totals at each separate rearrangement), it may appear probable that this group contains some abnormal bars, and it may also appear possible that some of the other large groups show an agreement through the averaging of bars showing wide differences among themselves.

Table XVIII gives some information on this point. Every group in Table XVII comprising more than 50 heats and containing less than 0.225 per cent. of carbon is subdivided so as to have only one-half the former variation in manganese. Thus, if a group comprised heats ranging from 0.4 to 0.49 per cent. of manganese, it is subdivided into one group ranging from 0.4 to 0.44 per cent., and another from 0.45 to 0.49 per cent. If the original group were an average of unlike units, it is probable that the fact would be made manifest by a wide difference between the two parts, but in no case is such a difference discernible.

In the case of the one group composed of 53 heats before mentioned, a more extended analysis is given in Table XVIII. It has been divided into 10 parts, the first containing only those heats that contained 0.4 per cent. of manganese, the second those with 0.41 per cent. of manganese, and so on. The number of heats in some of the subdivisions is small and complete regularity could hardly be expected, but in these 10 subdivisions the smallest difference between the strength as calculated by the

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TABLE XVIII.—*Subdivision of the Groups in Table XVII that contain over Fifty Heats, and are below* 0.225 *per cent. in carbon, with special subdivision of the one large group showing a difference of more than* 1,500 *lb. between the actual and calculated strength.*

Limits of Carbon.	Limits of Manganese.	Number of Heats.	Chemical Composition.			Ultimate Strength.		
			Carbon.	Phosphorus.	Manganese.	Actual Records.	By Formula.	Difference.
Per Cent.	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Pounds per Sq. Inch.	Pounds per Sq. Inch.	Pounds per Sq. Inch.
Acid steel :								
0.125 to 0.174	0.40 to 0.44	55	0.1459	0.0569	0.417	60,560	60,484	—76
	0.45 to 0.49	41	0.1477	0.0564	0.470	61,483	61,250	—233
Basic steel :								
0.020 to 0.074	0.10 to 0.14	56	0.0327	0.0073	0.120	45,674	44,748	—926
	0.15 to 0.19	9	0.0319	0.0071	0.159	45,458	44,666	—792
0.075 to 0.124	0.40 to 0.44	33	0.0961	0.0086	0.418	49,809	51,294	+1,485
	0.45 to 0.49	20	0.0946	0.0079	0.470	49,434	51,784	+2,350
	0.40	12	0.0963	0.0075	0.400	48,949	50,965	+2,016
	0.41	4	0.0888	0.0110	0.410	49,510	50,824	+1,314
	0.42	5	0.0946	0.0080	0.420	49,469	51,096	+1,627
	0.43	4	0.1012	0.0083	0.430	50,626	51,812	+1,186
	0.44	8	0.0980	0.0091	0.440	51,053	51,776	+723
	0.45	4	0.0870	0.0078	0.450	48,521	50,869	+2,348
	0.46	5	0.0922	0.0084	0.460	49,693	51,455	+1,762
	0.47	3	0.0833	0.0073	0.470	48,383	50,718	+2,335
	0.48	4	0.1135	0.0080	0.480	50,993	53,452	+2,459
	0.49	4	0.0948	0.0075	0.490	49,253	51,982	+2,729
0.125 to 0.174	0.40 to 0.44	32	0.1522	0.0114	0.418	55,495	56,129	+634
	0.45 to 0.49	32	0.1541	0.0114	0.473	56,265	57,102	+837
0.175 to 0.224	0.40 to 0.44	66	0.2036	0.0090	0.416	60,344	60,095	—249
	0.45 to 0.49	59	0.2046	0.0107	0.468	61,208	61,247	+39
	0.50 to 0.54	48	0.2063	0.0139	0.514	62,358	62,584	+226
	0.55 to 0.59	17	0.2049	0.0124	0.566	63,086	63,199	+113

formula and the strength as found by the testing-machine, is + 723 lb., and the greatest is + 2,729 lb., so that the deviation of this group from the general rule is not due to one or two abnormal bars. With this one exception, the cause of which remains unexplained, all the large groups show a difference of less than 1,500 lb. between the actual and the calculated strength, which is perhaps as close an agreement as could be expected.

A careful analysis was made to discover whether anything could be learned from the so-called errors. If, for instance, the groups of low carbon had shown a considerable and uni-

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form minus error and the groups of high carbon had uniformly shown a similar plus error, then it would be probable that the value of carbon was too high and the base too low. Investigation failed to show any regular law either for groups of high and low carbon, or for groups of high and low manganese. The one fact which appears to be true of both acid and basic steel is that the steels that are low in carbon and low in manganese are stronger than would be called for by the formula, and it seems probable that this is due to iron oxide.

The Value of Manganese.

From Table XVII may be obtained data which will corroborate the variable value assigned to manganese. The groups containing from 0.3 to 0.39 per cent. of manganese may be plotted, using as abscissas the percentage of carbon and as ordinates the ultimate strength. The groups containing from 0.4 to 0.49 per cent. of manganese furnish another line; those from 0.5 to 0.59 per cent. another; and those from 0.6 to 0.69 another. Owing to this subdivision many of the construction points in these lines represent only a small number of heats and they have therefore been combined by groups of three as before explained. The result is shown in Table XIX. If the phosphorus in each group were constant, no allowance would have to be made for it, but since it varies considerably, the ultimate strength has been calculated to zero phosphorus, as shown in the last column of the table and as plotted in Figs. 11 and 12.

Fig. 11 shows that the line representing from 0.4 to 0.49 per cent. of manganese is slightly above the base-line of 0.4 of manganese, which is the limit already determined, below which a decrease in manganese does not weaken steel. The number of heats containing less than 0.4 per cent. of manganese in acid steel is small and they are confined almost entirely to the two low-carbon groups, so that no line has been plotted for them, but if these two groups be put upon the diagram it will be found that they are as strong as though the manganese were higher. Above the limit of 0.4 per cent., each increase in manganese raises the strength, and not only this, but the angle made with the horizontal is greater as the content of carbon increases. The lines tend to converge at the left of the figure

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TABLE XIX.—*Classification of Groups in Table XVII According to Manganese and Combination in Units of Three.*

Kind of Steel and Composition.	Chemical Composition.			Ultimate Strength.	
	Carbon.	Phosphorus.	Manganese.	Actual Records.	Deducting Effect of Phosphorus.
	Per Cent.	Per Cent.	Per Cent.	Pounds per Square Inch.	Pounds per Square Inch.
Acid steel; Mn = 0.4 to 0.49 per cent. Line AA, Fig. 11.	0.1532	0.0569	0.441	61,840	56,180
	0.1691	0.0570	0.444	63,250	57,550
	0.2335	0.0548	0.456	69,963	64,483
	0.3127	0.0476	0.461	77,465	72,705
	0.3651	0.0412	0.462	82,238	78,118
	0.4004	0.0391	0.461	85,750	81,840
	0.4339	0.0366	0.464	89,060	85,400
	0.4738	0.0349	0.466	93,528	90,038
	0.5264	0.0325	0.470	98,860	95,610
Acid steel; Mn = 0.5 to 0.59 per cent. Line BB, Fig. 11.	0.1791	0.0584	0.520	65,590	59,750
	0.1951	0.0580	0.521	66,962	61,162
	0.2614	0.0525	0.533	73,985	68,735
	0.3305	0.0482	0.541	81,259	76,439
	0.3605	0.0452	0.541	84,250	79,730
	0.3938	0.0428	0.540	87,650	83,370
	0.4361	0.0394	0.537	92,030	88,090
	0.4679	0.0383	0.537	95,640	91,810
	0.5028	0.0387	0.533	99,800	95,930
Acid steel; Mn = 0.6 to 0.69 per cent. Line CC, Fig. 11.	0.3553	0.0463	0.619	85,940	81,310
	0.3793	0.0442	0.619	88,240	83,820
	0.4374	0.0392	0.619	95,280	91,360
	0.4716	0.0366	0.615	99,280	95,620
Basic steel; Mn = 0.3 to 0.39 per cent. Line AA, Fig. 12.	0.1131	0.0097	0.360	51,505	50,535
	0.1458	0.0098	0.363	54,107	53,127
	0.1989	0.0096	0.363	58,620	57,660
	0.2427	0.0089	0.366	61,900	61,010
	0.2678	0.0089	0.365	63,860	62,970
	0.3132	0.0104	0.363	67,397	66,357
Basic steel; Mn = 0.4 to 0.49 per cent. Line BB, Fig. 12.	0.1127	0.0098	0.441	51,920	50,940
	0.1668	0.0099	0.441	57,040	56,050
	0.2129	0.0104	0.446	61,340	60,300
	0.2415	0.0101	0.445	63,775	62,765
	0.2689	0.0101	0.446	65,955	64,945
	0.3065	0.0096	0.440	68,807	67,847
Basic steel; Mn = 0.5 to 0.59 per cent. Line CC, Fig. 12.	0.1240	0.0112	0.534	54,360	53,240
	0.1745	0.0125	0.531	59,315	58,065
	0.2163	0.0133	0.529	63,210	61,880
	0.2390	0.0133	0.530	65,315	63,985
	0.2667	0.0130	0.532	67,560	66,260
	0.3018	0.0125	0.536	70,495	69,245
Basic steel: Mn = 0.6 to 0.69 per cent. Line DD, Fig. 12.	0.1887	0.0154	0.622	61,864	60,324
	0.2192	0.0151	0.622	64,373	62,863
	0.2347	0.0146	0.621	66,020	64,560
	0.2695	0.0139	0.624	69,465	68,075
	0.3223	0.0149	0.623	76,278	74,788

at about 40,000 lb., which has been found to be the base for acid steel, and they tend to spread as they go to the right, which shows that the strengthening effect of manganese increases as

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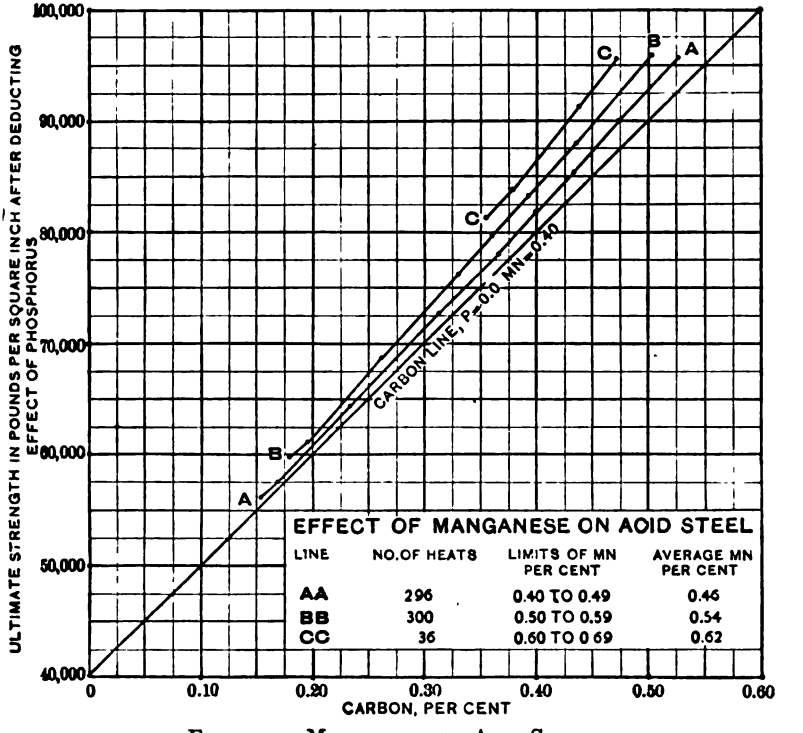
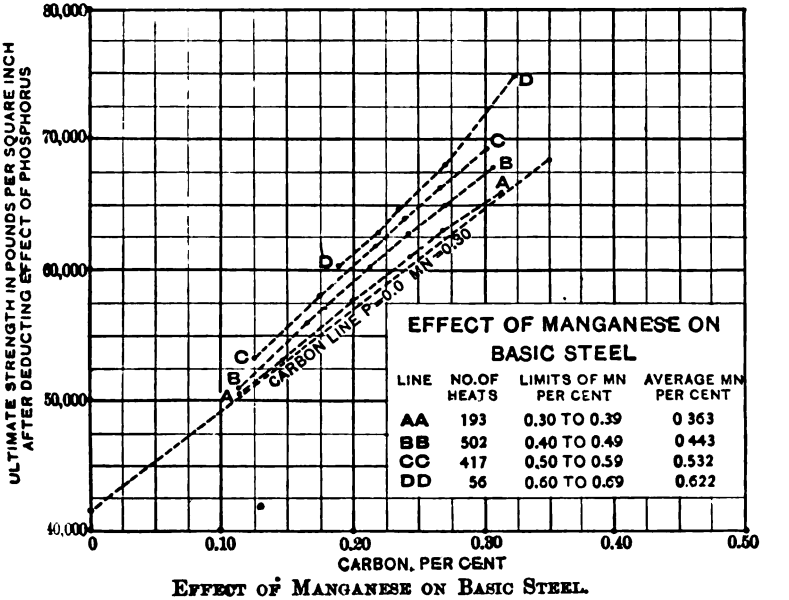
FIG. 11.



EFFECT OF MANGANESE ON ACID STEEL.

FIG. 12.



EFFECT OF MANGANESE ON BASIC STEEL.

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the content of carbon increases, and it is possible to estimate the effect by noting at what point a prolongation of these lines will cut a given ordinate, say the ordinate of 0.60 per cent. of carbon. It is estimated that the line AA will cut this ordinate at 103,000 lb. and it is known that according to the formula for acid steel the strength of steel of 0.60 per cent. of carbon and 0.40 per cent. of manganese would be 100,000 pounds. There is therefore an increase in strength of 3,000 lb. due to the amount of manganese in excess of 0.40 per cent. In Table XIX it will be seen that the average manganese of the line AA is about 0.46 per cent., so that the strengthening effect of manganese in a steel of 0.60 per cent. of carbon, as shown by this one line, is 500 lb. for each 0.01 per cent. In a similar way the line BB gives a value of 429 lb., and the line CC, 454 pounds.

The variation in values does not represent any law, but arises from determinative errors and the results should therefore be averaged. The line AA represents 296 heats; BB, 300 heats; and CC, 36 heats, so that the true average is 464 lb. for the influence of 0.01 per cent. of manganese upon a steel of 0.60 per cent. of carbon. In Table VII the corresponding figure was 480 pounds.

The results from the basic steel are plotted in Fig. 12, the base being 0.30 per cent. of manganese instead of 0.40 per cent. as in acid metal. It will be evident that the same law holds good that the effect of manganese increases with higher carbons. An analysis according to the same method used in the acid steels gave a value for 0.01 per cent. of manganese of 200 lb. for a steel of 0.35 per cent. of carbon, when Table XIII calls for 234 pounds. In both the acid and the basic steels the agreement is all that could be expected, so that it would seem that in the steels under consideration the manganese has a value varying with the amount of carbon present.

Conclusions.

Carbon.—In acid steel each 0.01 per cent. of carbon strengthens steel by 1,000 lb. per sq. in. when the carbon is determined by combustion. The strengthening effect is 1,140 lb. for each 0.01 per cent. as determined by color, owing to the fact that the color-test does not determine all the carbon present.

In basic steel each 0.01 per cent. of carbon strengthens steel

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by 770 lb. per sq. in. when the carbon is determined by combustion. The strengthening effect is 820 lb. for each 0.01 per cent. as determined by color.

Phosphorus.—Each 0.01 per cent. of phosphorus strengthens steel by 1,000 lb. per sq. in.

Manganese.—Each 0.01 per cent. of manganese has a strengthening effect upon steel, and the effect is greater as the content of carbon increases. Below a certain content of manganese the effect is complicated by some disturbing condition, probably iron oxide, so that a decrease in manganese in very low-carbon steels is accompanied by an increase in strength. In acid steel each increase of 0.01 per cent. of manganese above 0.4 per cent. raises the strength of acid steel an amount varying from 80 lb. in a metal containing 0.1 per cent. of carbon to 400 lb. in a metal containing 0.4 per cent. of carbon. In basic steel each increase above 0.3 per cent. raises the strength an amount varying from 130 lb. in a metal containing 0.1 per cent. of carbon, to 250 lb. in a metal containing 0.4 per cent. of carbon.

Sulphur.—The effect of sulphur on the strength of acid and of basic steel is very small.

Formulæ.—From the foregoing results, the following formulæ may be written, in which C = 0.01 per cent. of carbon, P = 0.01 per cent. of phosphorus, Mn = 0.01 per cent. of manganese, R = a variable to allow for heat-treatment, and the answer is the ultimate strength in pounds per square inch. The coefficient of manganese in acid steel, called x, is the value given in Table VII, and applies only to contents above 0.4 per cent. The value of manganese in basic steel, called y, is the value given in Table XIII, and applies to contents above 0.3 per cent.

Formula for acid steel, carbon by combustion:

$$40{,}000 + 1{,}000\,C + 1{,}000\,P + x\,Mn + R = \text{Ultimate Strength.}$$

Formula for acid steel, carbon by color:

$$39{,}800 + 1{,}140\,C + 1{,}000\,P + x\,Mn + R = \text{Ultimate Strength.}$$

Formula for basic steel, carbon by combustion:

$$41{,}500 + 770\,C + 1{,}000\,P + y\,Mn + R = \text{Ultimate Strength.}$$

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Formula for basic steel, carbon by color:

$$42{,}000 + 820\ C + 1{,}000\ P + y\ Mn + R = \text{Ultimate Strength.}$$

I am indebted to my brother, J. W. Campbell, of the Pennsylvania Steel Co., for collecting the data and assisting in the arithmetical work involved in this paper.

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SUBJECT TO REVISION

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Gold-Mines of the San Pedro District, Cerro de San Pedro, State of San Luis Potosi, Mexico.

BY GEORGE A. LAIRD, SAN LUIS POTOSI, MEXICO.

(Lake Superior Meeting, September, 1904.)

I. Introduction.

This old and once famous district played, through its enormous production of silver and gold, an important part in the history of the State of San Luis Potosi. According to a pamphlet prepared under the direction of the Governor of the State, Sr. Don Blas Escontria, and presented to the visiting members of the Institute during the Mexican meeting of 1901, this district is situated 20 km. east of the city of San Luis Potosi, in a range of mountains of moderate height, extending 135 km. N. and S., and 16 km. E. and W.; and the mineralized area hitherto developed is about 600,000 sq. m. The history of the district, as given by the record dated 1778, and formerly kept in the San Francisco convent, says that it was discovered in 1575 by Sr. Don Pedro Irarte, and adds that the city of San Luis Rey de Francisco del Potosi received its name (in allusion to the famous Potosi mines in Bolivia) by reason of the remarkable productiveness of its gold- and silver-mines in the first period of their exploration, during which they paid to the Crown, as "quintos" (the fifth part of the product), $97,000,000, as appears from the accounts of the royal treasury. The amount of the *quientos* paid during the "bonanza period" of 80 years was $72,000,000. The larger sum first mentioned probably represents a longer series of years. But either will suffice to indicate the great wealth of the mines.

In 1663, the decadence of the industry began with the caving-in of several mines, concerning which the record of 1778, already cited, says, in substance, that on Good Friday one of the owners in the enterprise, in view of the unfavorable prospect of a lawsuit in which he was engaged, set fire to the timbers supporting the underground galleries and workings of the San

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Cristobal mine (in which nobody was working on that day), thus causing the caving-in of the Cristobal, Santo Domingo, San Pedro and adjacent mines.

The principal rich ore-deposits being in this ground, attempts have been made since 1760 to open an adit which would reach them. This adit, called "El Rey," was a failure, being run on a level too high for its purpose.

About 1816, the Socavon de Victoria (Victoria tunnel) was commenced; and about 1870 the present owners secured possession of the greater part of the mineralized ground, now bearing the name La Victoria y Anexas. Since that date, work has been continuously prosecuted, although this was done at a loss until 1895, when the erection of smelting-works in the city of San Luis Potosi made the enterprise profitable.

The first systematic attempt to survey and map the mines was begun in 1901 by Mr. Frederick W. Taylor, then general manager. Since then, many of the accessible workings have been surveyed by the engineers of La Compañia Metalurgica Mexicana, which operates under lease a part of the mines, and by the present administration of *La Victoria y Anexas.*

A topographic map of the mines in the San Pedro district is given in Fig. 1.

At the present time 100 openings exist—many more being lost and hidden. The work is carried on by three companies: El Barreno y Anexas, San Jose de Cocinera, and La Victoria y Anexas, the last-named owning four-fifths of the territory. The mines of the two former companies are inaccessible to the writer, and therefore can be spoken of in a general way only.

The territory below the level of the Victoria tunnel is under lease to the Compañia Metalurgica Mexicana.

The Guadalupe and Abundancia tunnels are operated by the Victoria company, ores being mined from the San Pedro El Alto, San Cayetano, and Guadalupe mines through the former, and from the Abundancia through the latter. This company is producing about 75 tons of ore daily.

All other mines are operated by "camperos," or short-time lessees, and are largely unsurveyed, since, by reason of frequent cavings and of promiscuous stoping and gophering, a map would be too complicated and expensive to have a value commensurate with its cost.

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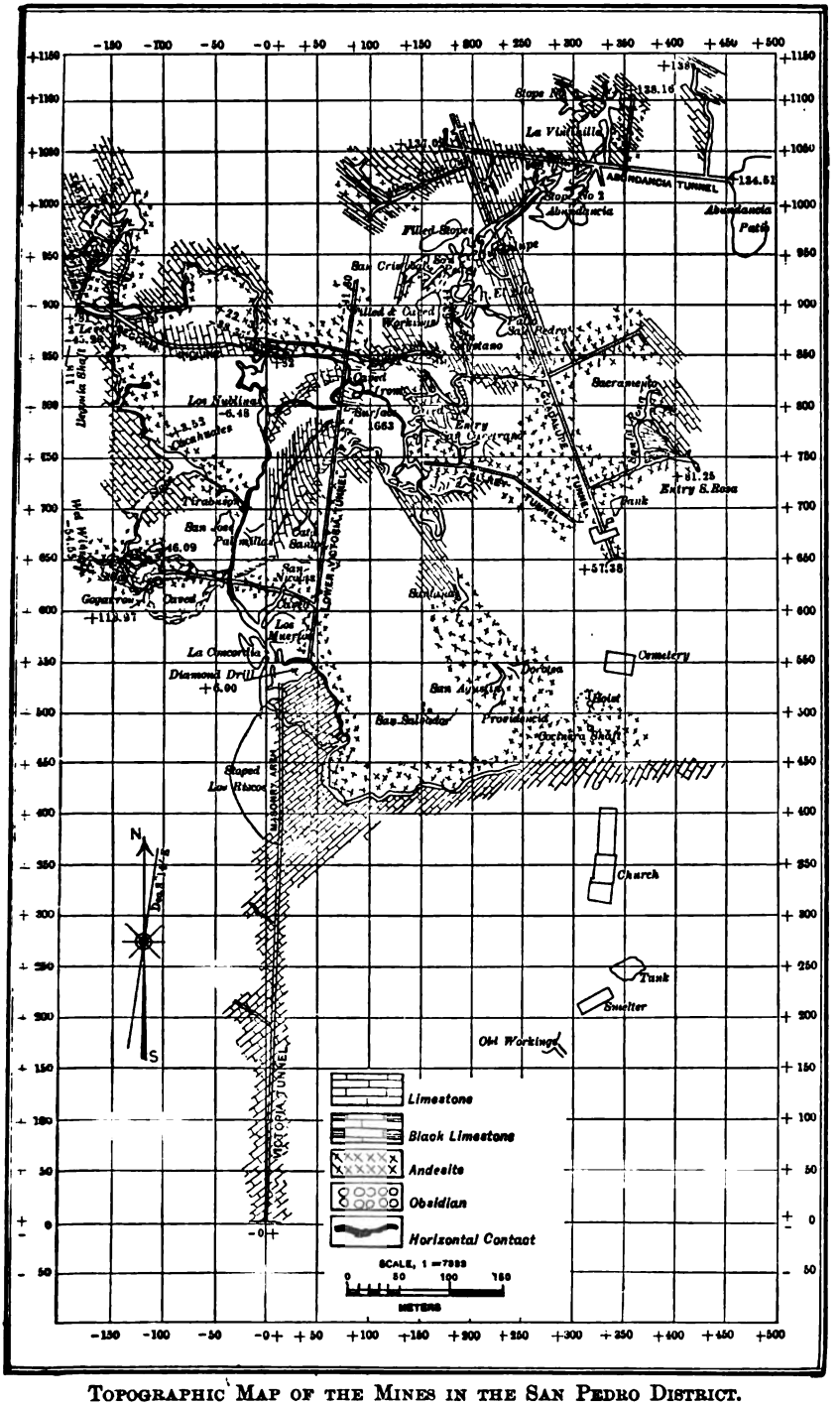
FIG. 1.



TOPOGRAPHIC MAP OF THE MINES IN THE SAN PEDRO DISTRICT.

[3]

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II. Geology.

The geological features of the district comprise a base of andesite, overlain by limestone, which is capped on some of the highest summits by rhyolite.

The rhyolite is barren. Ores occur on the limestone-andesite contact, or in the limestone near the andesite. The latter contains small secondary veins, but never has been found to carry paying quantities of mineral, although high assay-results have frequently been obtained from specimens.

The contacts and contact ore-bodies occur at every conceivable angle, but always exhibit, when explored, a hanging-wall of limestone and a foot-wall of limestone or andesite.

The ancient workings, opened at the outcrops, followed the veins to a maximum vertical depth of 250 meters. Several tunnels were driven during the 19th century, namely: the Victoria in 1818; the Abundancia and Guadalupe in 1890. In 1903, a new tunnel, the Dorotea, was commenced, but is now suspended, by reason of questions concerning surface-rights. One shaft, the Begonia, abandoned at the depth of 250 m., somewhere between 1873 and 1885, was re-opened in 1903.

The Victoria tunnel, after having been driven 500 m., encountered the caved ground of the disaster of 1663; and a shaft was sunk 50 m., and the lower Victoria tunnel was driven 400 m., presumably with the intention of cutting the Begonia shaft; but having missed that shaft by 250 m., in consequence of a mistake in direction, was long ago abandoned.

The collar of the Victoria shaft is immediately below the limestone-contact, and all its workings are in the eruptive rock. A diamond-drill hole sunk 450 ft. below the limestone failed to cut through to any other sedimentary rock.

The mine of the Cocinera company, operated through a 250-m. shaft, is near a contact, and shows limestone at bottom. The workings on the contact show the limestone to be dipping towards the drill-hole, but I doubt its continuance in that dip to the depth necessary to bring it 500 m. below the level of the Victoria tunnel.

The contacts in the district are clean-cut; and the limestone gives no evidence of metamorphism.

[1] Geology of the San Pedro District, by G. I. Finlay, Ph.D., *School of Mines Quarterly*, New York, November, 1903.

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The extent of the andesite has not been determined, and the whole of the territory now known to be mineralized is included in the relatively small area controlled by the three companies already mentioned.

III. The Ores.

The ores, which differ widely in various parts of the territory, may be grouped in seven classes, as follows:

Table I.—*Classification of Ores.*

Class.		Gold.	Silver.	Lead.	Silica.	Lime.	Iron.	Remarks.
				Per Cent.				
1. Hematites.	A	High.	Low.	None.	.....	.....	.....	.....
	B	Low.	High.	None.	.....	.....	.....	.....
	C	High.	Low.	5 to 15	.....	.....	.....	.....
	D	Low.	High.	5 to 35	.....	.....	.....	.....
2. Carbonates.	A	High.	Low.	5	High.	High.	Low.	.....
	B	Low.	High.	25	High.	Low.	Low.	.....
3. Residual cherts.	A	High.	Low.	1 to 25	High.	.....	Low.	.....
	B	Low.	High.	1 to 25	High.	.....	Low.	.....
4. Secondary sulphides.		Low.	High.	.....	.....	.....	.....	Occurring as galena or copper and iron pyrites.
5. Secondary copper-ores.		.....	.....	.....	.....	.....	.....	Carbonates and sulphides, not in paying quantity.
6. Lime-ores.		High.	Low.	None.	Low.[a]	High.[b]	Low.[c]	[a] 25 per cent., [b] 40 per cent., [c] 3 per cent.
7. Chlorides.		Low.	High.	None.	High.	None.	Low.	In andesite.

Note.—In this classification, the terms "high" and "low" apply to values, not weights. Thus 20 g. of gold and 200 g. of silver per ton would be "high" gold and "low" silver, representing in Mexican currency about $24 gold and $7 silver. On the other hand, 6 g. of gold and 300 g. silver ($6 and $10.50 respectively) would be low gold and high silver.

It will be seen that the same class of ore is never high in both gold and silver. When the different classes occur side by side, they have entirely different characteristics, and are easily distinguished; but ores of the two extremes in gold and silver, coming from different mines, or different parts of the same mine, are often so nearly alike as to make determination, by hand-glass or sorting, impossible.

The gold occurs native; the silver, associated with the lead or as a chloride; the lead (except in the secondary sulphides), as a carbonate; and the iron as hard blue and soft red hematite. Silica appears in the ores as chert, rarely as quartz,

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sometimes as agate; and lime, as the country-rock, or redeposited as calcite crystals.

IV. The Mines.

The large surface-area of the caved-in portion of the mines indicates stopes of enormous size, which are now inaccessible. Work is now going on with the intention of cutting into the caved portion in search of ores which, left as unprofitable by the ancients, would now be ranked, for modern methods of mining and smelting, as of comparatively high grade.

1. *San Pedro El Alto.*

Probably the oldest of the workings now open are those of San Pedro el Alto. Here siliceous ores occurring at the surface were followed down; and the area stoped, together with the small pockets of ore remaining, indicates that the greater part of the ancient product consisted of hematite and soft lead carbonates.

The ore-bodies must have been extremely rich and large. Free gold has been found in kidneys of from 10 to 12 kilos, and ores carrying 10 kilos gold and 1.5 kilos silver per ton were extracted as late as 1902.

Two classes of ores are now being mined in the large stopes near the surface: hematites high in gold, and low in silver and lead (class C in the foregoing table); and hard carbonates, or residual cherts, low in gold, high in silver, and carrying 25 per cent. of lead.

A hard, dark blue hematite, occurring in veins through the softer hematites, frequently carries free gold to a high value; but when it does not carry free gold which is visible to the naked eye, it is almost worthless, being practically barren of both gold or silver.

In depth, the ores occur as: neutral, free from lead; neutral, with 25 per cent. of lead; hematites, free from lead; and siliceous ores, free from lead. They are all low in gold and high in silver. The ore-bodies almost always occur in pairs, *i.e.*, two large bodies are separated by limestone or by low-grade and unprofitable mineralized matter. The softest material, and probably also the richest of the hard ores discovered by the removal of the soft streaks, have been stoped away.

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There are indications that the ancients were unable to handle the chert profitably, by reason of its hardness, although much remaining in the mines has been vitrified by the fires with which they sought to soften it. Grinding it by arrastra must also have been difficult.

The fillings of the stopes contain both fine and coarse ore, of good grade under existing treatment-conditions; and much which is now unprofitable may prove valuable at a later period.

From the surface to the Guadalupe tunnel-level (over 100 m. vertical) there are three pairs of large stopes. Below this level there are other ancient workings, which will be opened hereafter. At present any increased production from this point could not be handled.

The surface-ores now remaining are the richest, diminishing in value to the Guadalupe tunnel, where the lower workings connect and form a "U," the opposite end being worked through the Abundancia tunnel. The ore-bodies at that end are entirely different in character from the San Pedro deposits. Although about the same as to gold-content, they are lower in silver; always carry lead; are neutral as to silica and iron; and, near the surface, frequently contain as much as 15 per cent. of lime.

At Cueva Colorado, a surface-opening from the upper stope of San Pedro and about 20 m. higher than the mine-entry, ores are now produced containing about 100 g. gold; 100 g. silver per ton, and 3.5 per cent. of lead, 20 per cent. of iron and 30 per cent. of silica. The old fillings, beneath these backs, contain 30 g. gold, 160 g. silver, 5 per cent. of lead, 20 per cent. of iron, and 30 per cent. of silica.

Small stringers in the limestone hanging-wall give 6 g. gold, and 1,600 g. silver, per ton, and 70 per cent. of lead.

The northerly side of the stope contained a small 18-in. stringer of ore between limestone walls, which, being followed, led to a stope (the former entry to which has not been discovered after ten months' work), called *Salon de los Mojoneras*, from the fact that it contained a number of monuments of loosely piled stones.

More than 1,000 tons of ore, the greater part of which has been chert hematite, have been removed without encountering walls.

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Various small veins or feeders were followed by the ancients, one of which led to "the sick stope," so called because the miners were poisoned by arsenic. The ores carried approximately 75 g. gold and 50 g. silver per ton, and 5 per cent. of lead, and 20 per cent. of arsenic. The general mass of this ore is chert, frequently resembling quartzite, carrying the lead and arsenic in veins. The gold is free and can easily be panned out, but lead and arsenic may have made it difficult of amalgamation, and thus caused it to be left. I doubt the theory that this was done on account of the effect of the arsenic upon the workmen.

Below these stopes is another pair, in which nothing remains to denote the character of the ores extracted; but the pillars existing between the two deposits are limestone, with veins of chert, carrying 15 g. of gold, and 250 g. of silver per metric ton, and 10.4 per cent. of lead, 10 per cent. of iron and 65 per cent. of silica.

The third pair, or practically a "double" pair, contain hematites, cherts and lead carbonates. The hematites are soft and contain about 5 g. of gold and 100 g. of silver per ton, and 2 per cent. of lead, 45 per cent. of iron, 20 per cent. of silica, and 5 per cent. of lime.

The open stopes from the crest of the hill to the Guadalupe tunnel show an extraction of about 600,000 cu. m.

At the point of union with the Abundancia workings there is a decided difference in the ores, which here carry 2 per cent. of antimony, and traces of arsenic.

The entire series of deposits are in the limestone or adjacent to the andesite, but the latter does not contain secondary veins at the point of contact, although stringers or hematite follow the line of contact for a short distance, on either side of the ore.

The general formation of the deposits in San Pedro el Alto comprises a hanging-wall of limestone, which frequently contains small kidneys of lead carbonates, carrying as much as 71 per cent. of lead, very low in gold and very high in silver. Below this, the ancients have left a hard *Guija* or residual chert, high in lead carbonates, low in gold and high in silver.

The hematites, which lie below the chert cap, are high in gold, low in lead and silver, and extend to the limestone footwall, which is practically barren.

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2. *Abundancia.*

In the Abundancia no chert exists: the silica is displaced by lime. The values are about the same with the exception of the hematites on the foot-wall, which are practically barren, and large bodies of low-grade ores in contact with the andesite, which, although containing lead and having the same general appearance, are of too low a grade in gold and silver to have a mining-value. These latter bodies, constituting another "pair" of deposits, differ greatly from those of the upper San Pedro. One of them is horizontal, while the other is vertical. I do not doubt that the two are connected, although this is not absolutely certain. The vertical deposit has produced thousands of tons: 30 g. of gold and 70 g. of silver, per ton, and 10.5 per cent. of lead, 15 of iron, 50 of silica and 1 per cent. each of arsenic and antimony.

The hanging-wall is limestone, the foot-wall andesite; and although there are many small veins leading to the stope, the general character of the ores is the same, varying in iron and silica from neutral to high excess of the latter.

In the horizontal stope, which is of recent development, the ores contain approximately: 20 g. of gold and 60 g. of silver, per ton, and 7.5 per cent. of lead, 25 of iron, 25 of silica, 8 of lime, and 1 per cent. each of antimony and arsenic. The hanging-wall is lime and the foot-wall a low-grade hematite, carrying 7 per cent. of lead, which rests on the andesite or lime.

A large boulder of limestone in the center of this stope was supposed at first to be a foot-wall, with a second deposit of hematite below. This may prove to be the case, as the carbonate ores lie on the limestone at the farthest point developed. The stope now measures approximately 40 by 20 by 10 m., with the back and one side to the country-rock. All ores are stoped to funnel-shaped mill-holes, which lead to levels driven from the main tunnel. At the farthest chute from the main tunnel, the formation is hematite capped by limestone, about 2 m. thick, which lies directly beneath the pay-ore.

From the Abundancia, a passage-way driven *pozo patilla*, or by the winze-level system (probably to keep the peons from falling more than a certain distance), has been connected with the Guadalupe and put in shape for the extraction of ores by that

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tunnel. A single ore-body, from the Guadalupe level to this connection, demonstrates that veins, at least, connect all the bodies from San Pedro el Alto to Abundancia. Development-work is being carried on with the expectation of encountering the companion of this single ore-body.

3. *San Cayetano.*

San Cayetano, another of the old mines, contains three classes of ores:—hematite, low in gold, high in silver, and high in lead; carbonates, low in gold, high in silver, and high in lead; and silicates, low in gold, high in silver, and carrying no lead.

Although the iron and the siliceous ores lie side by side, and the siliceous ores are very porous, they are comparatively little stained by the iron-ores, which in places are highly oxidized.

At the present time, the mine is being cleaned of waste and ore from below. Eight months of steady work has failed to open a passage-way from the Guadalupe to the surface. Much ore is extracted with the waste, and much remains which, owing to the loose condition of the fillings, cannot be extracted until everything is free above, which will be in six or eight months.

4. *Santo Domingo.*

The Santo Domingo is badly caved, and the ores extracted from the old fillings by *camperos* are high in gold, low in silver, high in lead and excess of insoluble material.

In these workings, occasional boulders of copper oxide are found, running as high as 50 per cent. of copper. The accompanying minerals are not discolored; and copper does not occur in paying quantities.

Here, as in San Cayetano, the effects of the cave-in of 1663 are plainly evident, many places in the passages being between large limestone boulders.

The Dorotea tunnel will cut the workings at approximately 100 m. depth; and much of the old filling can be profitably removed.

The deposits of the Abundancia, San Pedro el Alto, San Cayetano, and Santo Domingo, compose the principal district of iron-ores, with the exception of the Begonia shaft, which will be mentioned later.

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These four mines may be considered as a single series of deposits, comprising ores of three classes:

1. Hematites, both hard and soft, high in gold and low in silver; or low in gold and high in silver; and low or high in lead.

2. Hard and granulated cherts, high in gold and lead and low in silver; or high in silver and lead, and low in gold.

3. Hard or soft carbonates, high in gold and low in silver; high in silica and low in iron; and high to low in lead.

5. *Other Openings.*

On the opposite side of the mountain are the openings of Cata Santos, San Nicolas, Palmillas, Los Muertos, San Pedro el Bajo, and others.

The ores of these openings are widely different from those already described, and consist of residual cherts, carrying from 5 to 25 per cent. of lead as a carbonate, and high gold with low silver, or high silver and low gold; the lead evidently carries the silver; and the ores decrease in gold with increase in lead. They run as high as 2,500 g. of silver per ton. Iron and silica remain in about the same relative proportions; and an average would be about 18 per cent. excess of insoluble. Hematites rarely occur. The lead is very fine, and not easily distinguished without panning; and the ore-bodies, as developed, do not show any contact with porphyry.

Much of the surface is badly caved in. All surface-indications in the immediate vicinity have been followed; and nearly all of them have led to ore. Although in many cases the ores thus encountered have been considered as merely superficial, I believe that they continue to the andesite, the presence of which in depth is demonstrated by the Victoria tunnel and the Begonia shaft. These workings will be developed through the new tunnel, at a depth of approximately 75 m., there being fifteen old surface openings to be passed in 250 meters.

North of these mines is the Gogarron, which carries ores of all the varieties except copper-ores found in the district. An incline, with a general northerly direction was sunk on ore its entire length. Large pillars of all grades of ore demonstrate that a large tonnage was extracted. At 169 m. in vertical depth, masses of limestone from the cave-in of 1663 are found, showing that the old workings were more than superficial.

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There are passages from the lower Gogarron workings to Las Nublinas, a solitary stope, still remaining open, although all roadways are through badly caved ground. The existing conditions point to a sag in the limestone; which, although in close proximity, it lies flat, is here nearly vertical. In Las Nublinas, the ores now mined by leasers are siliceous lead carbonates, high in silver, low in gold, and carrying as high as 10 per cent. of antimony.

From Las Nublinas to the Victoria tunnel, the passage is on, or very near, the bed rock for its entire distance, and the sides are walled with "trinchas" or piled waste rock.

The andesite is in places slightly discolored with copper carbonates and contains veins of secondary sulphides, from 20 to 100 mm. in width. Bodies of ore have been removed from above, and some of good grade still remain, but will be inaccessible until reached by new levels.

6. *Victoria Tunnel.*

From the Victoria tunnel openings were driven, large enough to permit working in all directions. Ores were encountered in many of these, which have long since been abandoned.

At the collar of the interior shaft much stoping has been done, and is still going on. Thousands of tons have been removed; and in the whole Victoria district the stopes indicate an extraction of millions of tons.

The ores now mined are highly siliceous. At the surface, where the Barreno company is working, a large excess of insoluble matter over iron is present, although from 25 to 30 per cent. of lime is not unusual. The surface-ores, high in gold and low in silver, carry practically no lead; but, in depth, the general values fall to very low gold, and high silver, and carry 5 per cent. of lead.

Traces of copper carbonate are present; and a small quantity of cuprous ores has been mined, and the limestone carries secondary veins of chalcopyrite, galena, pyrite, and zinc blende, not in paying quantities.

In the lower Victoria tunnel, all of which is in andesite, there are numerous secondary veins from 20 to 100 mm. thick; and in the backs of this tunnel and the cross-cuts driven from it there are long threadlike crystals of alum. In the Pozo de

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Agua, the extreme western point of the company's workings, there is a small amount of obsidian, with feldspar crystals which contain copper, native and as carbonate.

From the Victoria tunnel, the Princessa workings follow the contact, encountering a number of large stopes, which contain *almagre*, a deep red oxide of iron, used by the natives in making paints.

Lead-ores, carrying an excess of insoluble matter, also occur; and a large stope, supposed to have produced a high grade of copper-ore, shows discolorations, but no other evidences that copper really did exist there.

These workings (with the exception of the Gogarron), like those first mentioned, although not so thoroughly demonstrated by connections, are evidently on the same series of deposits, but as far as known do not touch the first group. The largest stopes in the district are in the vicinity of the Victoria tunnel. Los Riscos and La Campana are the principal ones.

At the breast of the Victoria tunnel there is a small area of black limestone, carrying secondary veins of copper, iron, lead, and zinc sulphides.

This stratum, apparently stained black by vegetable matter, occurs in the Begonia shaft also, although its continuity is not evidenced by the other mines between these two points.

7. *Begonia Shaft.*

The Begonia shaft was abandoned about 1870, at a depth of 275 m., and reopened in 1903 by La Compañia Metalurgica Mexicana, which now operates it, extracting hematites, low in gold, high in silver, and carrying 30 per cent. excess of iron over silica.

The shaft has penetrated the andesite, but cross-cuts demonstrate the limestone to be continuous to the depth of more than 200 meters.

The lowest mine-level (30 m. below the deepest known lime), driven in andesite to catch the lime, has passed by 15 m., the point expected to show the contact; and I doubt its existence at that depth. If encountered, it will not necessarily indicate that the contact extends to great depth, since all the indications above point to a general horizontal, rather than vertical, formation.

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Here, as in the lower Victoria tunnel, there are no evidences of mineralization in the country-rock. All ores are found on or near the country-rock, and the general rule of lime hanging-wall and lime or andesite foot-wall holds good.

The deposits are evidently continuous from the Gogarron, similar in character, and of corresponding contents. Unlike those of the San Pedro el Alto group, they do not crop out at the surface. In my opinion, the shaft was sunk to cut the Gogarron contact.

The manway (an incline, together with *pozos y patillas*, or winzes and levels) shows evidences of much travel from below: the limestone in the hand- and foot-holds of the peones has been worn as smooth as glass.

A passage from the foot of the incline leads to San Cayetano, while one lower down leads to El Rey tunnel. This tunnel, originally driven to cut the caved ground, shows no regard for level, but is otherwise a beautiful piece of work, about 1.5 by 2.5 m. in size, and cut with gad and moil to a perfect arch in the back, with sides smooth and even. Evidently it was intended for pack-animals; but, after a few hundred meters of really fine work, the old methods were returned to, and a great deal of "hand-and-knee" work is necessary in order to get down to the Victoria tunnel level. In the lower workings, after crawling several hundred meters, one suddenly enters a level, about 50 m. long, which is as well finished as any level in the mine. How it happened to be driven is a mystery.

Much stoping has been done in the upper levels; but they are at present inaccessible.

V. Methods of Mining.

There are about 200 miles of workings, from which ores of many millions of dollars in value have been extracted, and every indication points to a future extraction of many thousands of tons of marketable ore.

How the connections between mines were made without plans, as they undoubtedly were, is unknown. Much of the work was done with the aid of fire in place of explosives. All the ore was packed to the surface by peons or burros; and, notwithstanding the low cost of labor (18 cents per day), the operating expenses must have been large.

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By reason of its cost, timber was not then, and is not now extensively used. Supports were, as now, dry walls of limestone, one exception being a masonry arch 100 m. in length, which occupied 4.5 years in construction, and now sustains the fillings above the Victoria tunnel.

Stagings were built of poles lashed together with leather thongs, or with ropes made of *ixtl* (the fiber of the *lechuguilla*). That drills were sometimes used is proved by one now in the possession of Mr. H. Sturgis, of San Pedro. It is made of iron, drawn to two points on the circumference in place of a bit.

In the Begonia shaft, a *malacate de sangre*, or whim, was employed, leather *botas*, or sacks, being used in place of buckets. This has been replaced with a complete steam hoisting-plant.

Much of the ore was brought to the surface by peons, from a depth of 250 m., through passages barely large enough to admit a man (the leasers still employ peons for this purpose), and all waste was left in the mines when possible. Much of this is now worth extracting; but the extraction is rendered difficult and costly by the fact that a large amount must be handled in order to procure a comparatively small quantity of pay-stuff, and by that lack of surveys of the stoped-out parts, many of which have been encountered in supposed virgin ground.

Much ore has been hidden by dishonest employees and leasers. I obtained more than 5,000 tons of ore from a filled stope, which the mine-foremen had declared to contain nothing but waste, and many hundreds of tons still remain, carrying less than 10 per cent. of unprofitable material.

The difficulties of working the property must be experienced in order to be appreciated. Carefully-planned exploration generally encounters "fillings," the extraction of which frequently causes complications. A 400-ft. chute, intended to reduce the cost of extraction from San Pedro, was set at an angle of 34 deg. It was known that the ore would not run at that angle, but the conditions rendered a steeper incline impracticable. Two days after the completion of the work (the bed being placed on stone-walls), the floor fell into an old stope. Everything having been repaired, a considerable quantity of ore was extracted, but in this work another old stope was cut; and as the surveys and angles of the walls showed no

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probability that this was the same as the first, the work of emptying it was begun. After four weeks, the entire lower part of the chute began settling; the spreaders and braces loosened; and the emptying of the stope has been stopped until a surface-connection can be made.

The Big Cave.

Since I began this paper the "big cave" has been penetrated. On Sept. 3, 1903, an old stope was encountered; but this is so common an occurrence, and so generally without special wider significance, that the possibility of a connection with the big cave was not considered, especially as this particular old stope was opened on the north side of the level we were driving in order to reach the cave, which we expected to find on the south side, 60 m. further on. But it turned out that we had struck a part of the big cave.

At the present time little can be said of this ground. The workings at the point of encounter contain badly-charred timbers, but it is not clear that they were in any way connected with the reported fire. The ores exposed are granulated and hard cherts on the andesite foot-wall, and carry much higher values than any of the other ores now under extraction. No hanging-wall has been exposed; but the fillings leave no doubt that it is limestone.

The cherts were evidently found to be too hard for extraction, or else not as rich as the soft iron-ores, of which small stringers remain. Partially fused rock still hangs on the walls; and ashes demonstrate that some of the chert was attacked by fire. About 10 tons of ore was found piled up, apparently for removal.

So far, our work has failed to disclose the entrance. The stope continues below the point cut by our level. The ground is not badly caved; but it is much crushed, and the air-currents through it indicate former mine-openings of more than ordinary size. Smoke from blasting rises, without entering into other parts of the mine; and our surveys give no clue to the hidden connections. An area large enough to admit 40 pairs of miners, all working on ore, has been cleared, with no indications that the entire ore-body has been exposed.

What will be found in this ground hereafter can only be con-

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jectured; but it is more than probable that whatever may be found will be of lower grade than the ore which has been removed.

VI. Source and Distribution of the Ores.

The fact that the andesite has been proved, although in but one known location, and that of extremely small area, to contain profitable values in silver chloride, and (in a number of cases) native silver, led me to sample the porphyry thoroughly; and on this examination I base the following conclusions:

The rhyolite is barren: no ores occur on the rhyolite-limestone contact.

All deposits at some point touch the andesite.

The andesite carries gold and silver at a general ratio of 1 g. of gold to 10 g. of silver; or 2 g. of gold to 20 g. of silver, per metric ton.

The andesite never contains visible gold or even moderately high assay-results in gold; and the ores carrying free gold (and sometimes gold crystals) never carry visible native silver, and very rarely silver chloride.

High-gold and high-silver never occur in the same ore.

It is my belief that the ore in these deposits can be directly attributed to the porphyry; and in support of this hypothesis I offer the following remarks.

The assay-returns of gold and silver from the andesite of various parts of the district were so similar, that Mr. Franklin A. Hall, the chemist at the mines, agreed with me in calling all the andesite samples "2 to 20 rock." Indeed, samples taken later from the diamond-drill core, 900 ft. below the Victoria tunnel-level, gave approximately the same contents of gold and silver. The calculations given by me in a semi-annual report to the company showed the quantities of gold and silver produced to be so nearly in the ratio of 1 to 10, that I extended the calculation to the 16,197 tons mentioned in Table II., and have since that included 20,000 tons more; these additional calculations confirming the ratio named.

That gold and silver do not exist in any known solution in the ratio of 1 to 10 is a point I shall not here discuss. Nor shall I try to explain how the precipitation from such a solution could be greater in some localities than in others; and yet, that the aggregate result from all localities should give the ratio of 1 to 10.

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In the case of deposits formed through magmatic segregation, I should expect that the mass and the magma should bear the same ratio in metallic composition, *i.e.*,—that a ton of material mechanically concentrated from the magma should bear the same ratio in metals as that concentrated by segregation.

An alteration, such as of a sulphide to an oxide, might later take place, causing an apparent enrichment, giving an increased value in gold and silver per ton, while the value per unit of volume remained the same; and a ton of pyrite or pyrrhotite from the magma showed equal in gold and silver value a ton from the segregated mass.

The San Pedro deposits present every appearance of having been formed by the action of water,—not by magmatic segregation, as I understand the term, and not by mechanical concentration; and the "missing link" is the solution which would have an equal affinity for both gold and silver, unless the deposits were formed mechanically.

The hypothesis that all the metallic contents have been leached from the magma being disproved by the remaining percentages which have caused this ratio to be noted, there can be no calculation concerning the area leached in order to produce the deposits, nor the percentage extracted by such leaching; since the badly oxidized and decomposed rock give practically the same results as the clean rock.

Doubtless, humid analyses would have shown somewhat different results; but all these determinations having been made by fire, the average is assumed to be sufficiently correct for comparison.

Five diamond-drill samples, taken at a depth of 900 ft., and covering a vertical distance of 20 ft., show that the ratio of gold to silver varies from 1 : 26 to 2.5 : 9. The average, as shown by the results of 20 crucible assays, each charged with 15 g. of pulp, gave a ratio of 2 : 19.4 or 1 : 9.7, which I consider practically the same as 1 to 10.

All the exposed andesite is blocky, with joints or slips, which would account for the percolation of water to a considerable distance. The upper portions of the deposits may have been enriched through erosion and infiltration.

Even the hardest and firmest of the andesite slacks rapidly

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upon exposure to the air; and the slips frequently contain calcite crystals at some distance from the lime.

The metallic values of consecutive shipments to the smelter, aggregating more than 16,000 tons, were calculated according to the assay-results accepted for settlements, and the ratio in grams of silver to gold for each of the various classes of ore was ascertained.

The total amount of ore, comprising about 200 lots, constituted the entire product for the period covered of the mines named in the following list:

Abundancia, San Pedro El Alto, Guadalupe (lead-ores), Guadalupe (iron-ores), Victoria (dry), Purchased from leasers (lead), (dry), total, 16,197.[376] tons.

Table II. shows the results obtained, together with the general averages.

TABLE II.—*Silver, Lead, Iron, Silica and Lime per Gram of Gold, According to Assays Accepted for Smelter's Settlements.*

Quantity.	Silver.	Lead.	Iron.	Silica.	Lime Carbonate.
Metric Tons.	Grams.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
950.847	0.005465	0.05018	2.5150	3.0920	0.04910
2,704.477	0.005757	0.04978	2.1920	3.0040	.....
47.801	0.014044	0.12426	2.1880	3.3770	.....
1,892.843	0.021020	0.00800	7.8220	5.4320	.....
546.589	0.008377	0.02583	1.0090	1.6290	0.02590
736.919	0.010900	0.00146	1.1320	1.9530	0.01260
121.018	0.043595	0.00261	4.5280	6.9550	0.13450
259.015	0.061000	0.00584	3.9850	8.0270	0.07470
735.921	0.013834	0.00179	1.0920	2.0640	.....
3,698.855	0.003888	0.04021	1.4887	2.0802	0.07153
2,588.805	0.016157	0.01950	1.0533	2.3803	0.01198
973.844	0.014875	0.04865	2.6387	4.0751	0.01245
776.240	0.006120	0.02012	0.8107	1.3968	0.03512
164.200	0.049022	0.04026	4.3643	4.6750	.....
16,197.[379] tons, resulting in a general average, per gram of gold, per metric ton, of 10.017 g. silver, and percentages as follows: lead, 0.3069; iron, 1.33; silica, 2.5438; and carbonate of lime, 0.31.					

The lead-contents are shown for such ores only as contained 5 per cent., which will show a slightly decreased amount; "dry" ores generally contain, approximately, 2 per cent. of lead. The table comprises only ores actually shipped to the smelter, and hence the iron-contents reported in it are probably higher than the average of the mass of the ore-deposits of this

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particular class, because ores with an excess of iron, even though low in silver and gold, can be sold to smelters when siliceous ores of equal value in those metals would not pay the cost of smelting, and consequently ores too low in iron are not shipped.

The conclusion already stated, that high values in gold and silver never occur in the same ore, has been deduced from the following observations:

Andesite, contains by weight 1 gold to 100 silver (only when silver occurs as a chloride, or native, as before stated).

Hematite, contains by weight 1 gold to 2 silver;

Lead Carbonates (29.9 per cent. of lead), contains by weight 1 gold to 4300 silver;

Chert, contains by weight 1 gold to 230 silver;

Cherty Hematite, contains by weight 10 gold to 1.6 silver.

The silver-content almost, but not quite, always exceed in weight the gold-content.

The ratio of 1 of gold to 10 of silver has been shown in but few instances by ore from a single working-breast.

Beginning with the ores lowest in gold, marketed during the period covered by Table I., individual lots have contained the quantities of metal shown in Table III.

TABLE III.—*Metallic Contents of Individual Lots of Ore.*

Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.	Remarks.
Grams.	Grams.	Per Cent.	
4.0	85	1.8	Ferrous.
4.2	337	4.8	Siliceous.
6.5	628	0.0	Andesite.
8.5	53	5.0	Siliceous.
12.5	117	10.5	Siliceous.
18.0	284	5.1	Siliceous.
18.0	502	2.0	Siliceous.
26.0	334	5.6	Neutral.
28.0	58	5.1	Neutral.
78.3	182	4.4	Siliceous.
10,430.0	1,600	11.5	Chert-hematite.

It should be observed that a number of these lots came from leasers, and may have been composed of small lots of high-gold, low-silver, or high-silver, low-gold ores mixed. Smaller lots, ranging from 1 to 20 tons each (and less likely to have been mixed), have given the results shown in Table IV.

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TABLE IV.—*Metallic Contents of Small Lots.*

Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.	Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.
Grams.	Grams.	Per Cent.	Grams.	Grams.	Per Cent.
2.0	1,538	25.5	50.0	1,100	7.1
6.0	25,800	29.9	60.0	120	11.3
10.0	3,943	21.4	98.0	69	4.0
12.3	714	20.7	100.0	101	57.7
30.0	30	6.0	100.0	2,140	2.0
35.0	45	7.4	103.0	247	37.7
35.0	55	9.6	120.0	204	1.1
35.0	16,000	41.6	164.0	176	1.2
46.0	64	7.6	286.0	160	9.2
48.5	2,825	14.6	522.0	232	....
49.0	417	2.8			

It will be seen that the high gold-results are accompanied by low silver, and the high silver by high lead and relatively low-gold. The highest gold-values were contained in a chert-hematite, that is interstratified chert and hematite.

Analysis of the ores shows them to range from 50 per cent. excess of iron to 80 per cent. excess of silica. A large part of the present production consists of ores carrying an excess of insoluble matter. As to the metallic value of each class of ore, I have found no general rule, with one exception, namely, that the high lead carbonates occurring in the limestone hanging-wall always carry high values in silver.

In the majority of cases, high-lead is accompanied with high-silver; yet Abundancia mine-samples have given the results shown in Table V.

TABLE V.—*Assays of Abundancia Samples.*

Gold. Quantity per Metric Ton.	Silver. Quantity per Metric Ton.	Lead.
Grams.	Grams.	Per Cent.
10	100	22.0
20	80	19.0
31	89	17.5
34	76	24.0
49	79	22.5

On the other hand, the andesite-ores, as well as others high in silver, carry no lead.

The lead, and possibly also a small proportion of the iron, may easily have come from the limestone, and would, in that case, I think, have no bearing on the gold- and silver-contents.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Zinc-Smelting Industry of the Middle West.

BY H. C. MEISTER, E.M., ST. LOUIS, MO.

(Lake Superior Meeting, September, 1904.)*

General History.

The zinc-smelting industry of the United States has grown very rapidly in recent years and bids fair to outrival that of all other countries in the future. On account of the geographical situation of the chief ore-supplies, the industry may be divided into two main groups, the eastern and the western.

The eastern and older group includes the works of Pennsylvania, New Jersey and Virginia, which practically control their own ore-supply. The western group includes the works of Illinois, Missouri, Kansas, Wisconsin and Indiana, which do not control their own ore-supply, but are compelled to buy the ore in the open market. It is for this reason that the western zinc-smelting industry has been undergoing continual changes since its inception 40 years ago.

During the early discoveries of lead- and zinc-ore in the lead- and zinc-districts of Wisconsin and northern Illinois, the zinc-ore (smithsonite or "bone" as the miner calls it) had no value and was considered waste; but later, when this ore was found in large quantities, it attracted the attention of metallurgists, which finally resulted, during the early sixties, in the building of the Matthiessen & Hegeler Zinc Works at LaSalle, Ill., the pioneer zinc-works of the West.

A few years later, zinc-ore was discovered associated with the lead-ore in Jefferson county, southeastern Missouri, which started the industry at Carondelet or South St. Louis. Still later, the discoveries of zinc-ore in the lead-ore region, first at Granby and then around Joplin, lead to the introduction of the industry in the southwest Missouri and Kansas districts.

At first the development of the industry was slow, but it progressed steadily and it is now one of the leading metallurgical industries of the middle West.

* Presented at the supplementary session of the Institute in St. Louis, Mo.

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As this paper deals solely with the local zinc-industry the western group only is described. For convenience this group may be subdivided into States according to the progress of the development of the industry, which formerly placed Illinois at the head of the list, then Missouri, and later Kansas and Indiana. This order is now changed and Kansas heads the list, with Illinois second and Missouri third.

In a general way it may be said that after the establishment of the industry at LaSalle, Ill., in the '60's, there were four distinct periods of activity and enlargement followed by corresponding periods of depression. During the early '70's three plants were built at Carondelet, Mo., but after the panic of 1873 business became bad and did not revive until 1879. Activity was renewed in the beginning of the '80's, and many works were built in Kansas and southwestern Missouri. About 1885 a period of depression set in, followed by a very active period of construction during the early '90's. After 1895 a period of dulness occurred, which lasted until 1898, when the activity was once more revived by the building of a number of large plants in the Kansas natural-gas district. Since that time development in this district has been so rapid that a shortage of ore soon occurred. In 1899 the ore-supply was insufficient, and the resultant keen competition caused the price of blende to advance to the unprecedented figure of $54 per ton. The gas-smelters having a strong advantage over the old coal-smelters in the cost of production, the severe competition soon caused the closing down of all the coal-smelters in the Kansas and southwestern Missouri districts. The zinc-ores mined in connection with western silver- and gold-ores, however, soon relieved the shortage, and for the last four years large quantities of blende have been shipped from Colorado to Kansas for treatment in the natural-gas plants.

Production of Metallic Zinc in the United States.

The following statistics[1] show the rapidity with which the production of spelter increased with the application of natural gas.

[1] *Mineral Resources of the United States* for 1892, U. S. Geological Survey.

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	Short Tons.		Short Tons.
1895,	76,505	1900,	113,859
1896,	70,933	1901,	130,174
1897,	89,397	1902,	147,541
1898,	107,768	1903,	145,670
1899,	121,246		

An addition of 10,000 short tons for the eastern and southern districts represents the total production of the entire country for the years quoted.

Roasting Zinc-Ore.

Most of the zinc-ore now mined in this district is zinc sulphide or blende, hence a preliminary roasting is necessary in order to get rid of the sulphur. The more thorough the elimination of the sulphur during the roasting, the better the chances are for the recovery of the zinc in the subsequent smelting-operation. Each unit of sulphur left in the roasted ore will hold back double its weight of zinc, and as no sulphur by-products (such as matte) are formed in the smelting-operation, the loss by sulphur means so much waste in the residues which cannot be recovered. A variety of mechanical furnaces are used for roasting, although the old style hand-rabbled reverberatory furnaces are still in use at many plants.

The Hegeler Acid-Furnace.—Many years ago the Hegeler muffle-type of mechanically-rabbled furnace was introduced at the works of Matthiessen & Hegeler, LaSalle, Ill., with entirely successful results, but owing to the high cost of construction and the comparatively high cost of maintenance and operation, it has never been introduced except at works designed to make sulphuric acid from the sulphurous acid-gases. At present the Hegeler furnace is used at the following works: Matthiessen & Hegeler Zinc Co., LaSalle, Ill.; Illinois Zinc Co., Peru, Ill.; United Zinc & Chemical Co., Iola, Kas.; and Grasselli Chemical Co., Cleveland, Ohio.

With the exception of a small plant of the works of the Mineral Point Zinc Co. using the contact-process for making sulphuric acid, the four companies above named are the only manufacturers of acid from blende in the United States, the chamber-process being in general use. The difficulty of finding a market for sulphuric acid in the West has no doubt pre-

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vented its more extended manufacture. The acid market, however, is gradually widening, and no doubt the manufacture of sulphuric acid from blende will become more common.

A new departure in this line of work has lately been made by the Grasselli Chemical Co., which manufactures sulphuric acid from Joplin blende at Cleveland, Ohio, and ships the roasted ore to West Virginia to be smelted with natural gas.

Mechanical Roasting Furnaces.—The roasting of zinc blende in mechanically-stirred reverberatory furnaces marked the first advance in the reverberatory furnace practice. The first successful furnace of this type was the Brown horse-shoe furnace, invented in 1892, and its working gave an entirely new idea to inventors. Furnaces embodying the Brown invention sprang up like mushrooms all over the United States. Among the first to enter the field being the Pearce and the Ropp types, but litigation soon stopped their general introduction.

Compared with the ordinary hand-rabbled furnace, mechanical furnaces effect a considerable saving in labor and yield a better and more uniform desulphurization of the ore roasted.

Mechanically-stirred furnaces of the Brown type are in use at the works of the Edgar Zinc Co., at Carondelet, Mo., and Cherryvale, Kas., and at the plant of the Collinsville Zinc Co. at Collinsville, Ill.

Recently, a number of mechanically-stirred furnaces of the Ropp type were in use at the plant of the Lanyon Zinc Co., at Iola, Kas., but these had to be changed to the Cappeau type, in order to avoid an infringement of the Brown patents, the Ropp furnaces having been declared an infringement of Brown's patents.

Smelting Zinc-Ore.

There has been no radical change in the method of smelting zinc-ore. The old style of distillation is still in use, all attempts at direct smelting having resulted in absolute failure.

The Belgian type of furnace is universally used throughout the middle West, either direct-fired or gas-fired, the latter with or without heat-recuperation.

Hegeler "Blow"-Furnace.—The first attempt at gas-firing was made by Matthiessen & Hegeler and resulted in the development of the Hegeler "blow"-furnace. Mr. Hegeler found that by firing with gas, the length of the furnace could be

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greatly increased, all that was necessary being to introduce the quantity of gas required to heat the whole furnace and burn it gradually by introducing air at intervals. The Hegeler furnace, as now built, consists of a long rectangular chamber having retorts placed four rows high, and in some cases six; the gases from the producers enter at one end and are discharged at the other. Air for combustion is introduced through iron pipes entering the front of the furnace at various places alongside of the retorts. Both air and gas enter under pressure produced by a positive blast. The producers are operated with a hot top and are placed close to the furnace. No recuperation or regeneration is attempted in the furnace, but the hot gases escaping are used for the generation of steam for power-purposes. These furnaces have been built in size varying from 224 to 504 retorts to the furnace, or from 448 to 1,008 retorts to the block.

Siemens Furnace.—The next attempt at gas-firing was made by the Illinois Zinc Co., at Peru, Ill. This company adopted the Siemens regenerative furnace, built on the well-known principle of the Siemens regenerative system. The Siemens producers, however, proved inefficient and were later replaced by steam-blast producers. The furnaces were originally built with large elliptical retorts, arranged in 3 rows, but lately they have been replaced with circular retorts placed 4 rows high. The latest improvement on these furnaces has been the Neureuther patent, which consists in extending the air- and gas-channels partly up into the center wall, by which both air and gas are admitted into the furnace-chamber at a higher point than was usual. By this means it is claimed that the furnace can be built 5 rows high without detracting from its efficiency. It is also claimed that the furnace is heated more evenly on account of the shorter reach of the flame.

In 1882 the Rich Hill Zinc Works at Rich Hill, Mo., and the Granby Mining & Smelting Co., at Pittsburg, Kas., each built furnaces of the Siemens regeneration principle, which were failures, and were abandoned later. The Rich Hill furnace was built exactly like those of the Illinois Zinc Co., having large elliptical retorts, in three tiers, while the Granby furnace was built with small circular retorts arranged in four tiers.

The failure of the Rich Hill furnace was partly due to mis-

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management and partly to faulty construction. From my own observation the elliptical retorts were entirely too large for the heavy Joplin ore, and the furnace could not be worked "dry" in the required time; consequently the percentage of yield was poor. The Siemens producers required the use of a better grade of fuel which, at that time, was at least five times as expensive as the slack coal burned in the direct-fired furnaces. The most serious mistake, however, was in the arrangement of the gas- and air-channels underneath the checker-work. These channels, not being accessible during the time the furnace was in operation, soon became clogged with zinc oxide which had escaped from leaky retorts. So serious was the interference that an irregularity in the working of the furnace could be detected a few months after the furnace had been put in operation. This clogging caused frequent shut-downs, and instead of having produced a furnace with long campaigns and small repairs, it gave short campaigns and frequent and extensive repairs. In the Granby furnace the circular retorts were better adapted to the work, but the same defects of construction existed as in the Rich Hill furnace and the results were the same.

The high cost of construction of the regenerative furnace and the inability to use the common slack-coal of the district, together with the poor work accomplished on account of faulty construction, soon brought the improved furnace into bad repute; and so prejudicial was the result that no more gas-furnaces were constructed, everybody reverting to the old style of direct-fired furnaces.

Direct-Fired Furnace.—If it had not been for the local cheap slack-coal, other attempts at gas-firing would no doubt have been made, but slack-coal could not be burned to advantage in independent producers, on account of its inferior quality; hence the direct-fired furnace came into general favor. These furnaces were cheap to construct and gave, as a rule, good average results, although the consumption of coal was large and the labor was excessive.

The first direct-fired furnaces built in Kansas had square fronts, and contained 7 rows of 14 retorts each, which amounted to 98 retorts to the furnace, or 196 retorts to the block. Later, the size of the furnace was increased to 16 retorts in a row, set

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7 high, corresponding to 112 to the furnace, or 224 to the block, which remained the accepted type until the building of coal-furnaces ceased on account of the extensive discovery of natural gas in Kansas. In 1892 the natural-gas district of Indiana attracted attention, and a plant was built at Marion, followed by one at Ingalls and one at Upland.

Natural-Gas Furnace.—The application of natural gas to the zinc-distillation furnace, however, was not as simple as was at first supposed, and the first attempts for this purpose caused a great deal of worry and anxiety to the operators. The natural-draft furnaces of Indiana, in which a mixture of air and gas, on the principle of the Bunsen burner, was introduced through openings in the front or back of the furnace direct into the furnace-chamber, worked very irregularly. With the narrow channels of the zinc furnace, it seemed impossible to distribute and control the heat properly, and frequent "butchers" or break-downs of the retorts occurred with a correspondingly heavy loss of metal. Consequently, when the industry was started in the natural-gas district of Kansas in 1897, the first thing that suggested itself was to improve the distillation furnace and the method of using the gas. Natural draft was discarded in favor of positive blast. In fact, the Hegeler blow-furnace was copied as near as conditions permitted. In the original form of the natural-gas-fired distilling-furnace, the gas was introduced at one end, in the same manner as in the producer-gas-fired furnace, and the heat regulated by the introduction of air at various places. This arrangement, however, was not satisfactory, for the reason that natural gas, having a high calorific value, was not completely burned in the furnace by the air introduced for this purpose, and, as a result of the incomplete combustion, large quantities of carbon were deposited in the spaces between the retorts which clogged the furnace. In order to insure the complete combustion of the natural gas, and thereby overcome the deposition of carbon, more air was forced in, which, however, caused the furnace to become overheated. A careful study of the requirements indicated a change in the method of introducing the gas and air, so that they would enter jointly at various places through the front. A furnace was then designed which proved satisfactory, but the regulation of the heat of this form of furnace remained a difficult task, and, un-

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less watched very closely, irregularities and frequent breakdowns occur.

The peculiar heat-requirements of a zinc-furnace differ from most other metallurgical furnaces insofar that there is no large space in which to make use of radiant heat. On account of the narrow channels between the numerous rows of retorts the heat must be supplied almost entirely by conduction, which makes it necessary to surround the retorts continually with a flame having the required temperature. As the retorts are easily destroyed by over-heating, an excess of heat must be carefully avoided. The strong cutting-action of a blow-pipe flame, such as is produced in the blow-furnace, easily causes overheating and a consequent destruction of the retorts. It also tends to an early destruction of the lining of the furnace, causing frequent and expensive repairs.

The blow-furnace was first built in sizes of 5 rows of 60 retorts (300 retorts) to the side. Later, the combustion-chamber built at one end was discarded and the space was utilized by adding 20 additional retorts, so that now the accepted size is 5 rows of 64 retorts (320) to the furnace, or 640 to the block. Lately, the tendency has been to limit the number of rows to four, with a corresponding increase in length of the furnace in order to hold the same number of retorts, an arrangement which makes the furnace long, and in some works they have been divided into two transverse sections, placed in tandem in the same building. By working both sections as one furnace, the same economy of labor is retained. While the labor-requirements on these large furnaces is considerably less than on the small direct-fired furnaces, it is also more trying. The men being more exposed to the heat, which causes considerable trouble at times during the hot summer months. In order to avoid the trouble, the Edgar Zinc Co., at Cherryvale, limited each furnace to 5 rows of 20 retorts (100 retorts), or 200 to the block. By placing three blocks in one building, each separated from the other by a 12-ft. space, and operating the three small furnaces as one large one, the same economy of labor is obtained without having the men exposed to the heat as much as they would be with the single large furnaces.

These furnaces are also constructed to operate with natural draft and have proven quite successful. By admitting air and

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gas some distance below the lower row of retorts, primary combustion takes place before the gases reach the furnace-chamber proper; and by admitting more air, a secondary combustion takes place in the furnace-chamber, causing a very uniform heat without the deposition of carbon. Thus the cutting-action of the blow-pipe flame is entirely avoided, and repairs have been reduced to a minimum, which allows a long campaign of operation.

Lately, the coal-fired smelting-plant at Girard was removed to Chanute, Kas., to be operated with natural gas. The furnaces are arranged similar to those at Cherryvale, but each furnace has only 80 retorts, or 160 to the block, and the four blocks are placed in one building with each side operated as one furnace. These furnaces are also run by natural draft, similar to those at Cherryvale. Recently, a furnace was constructed at the works of the Prime Western Zinc Co., at Gas City, Kas., using the recuperative principle of the Converse and De Saulles furnace. This furnace is a patented one which was developed at the plant of the New Jersey Zinc Co., at Palmerton, Pa.

The Table I., which may not be strictly accurate, shows the changes that have taken place and the present condition of the zinc-industry.

Smelting in Colorado.—During the past few years the utilization of zinc-blende found associated with silver- and gold-ore in Colorado gave rise to the zinc-smelting industry at Pueblo, Colo. The United States Zinc Co., a branch of the American Smelting & Refining Co., built a plant at Pueblo, during 1903, which is said to be in successful operation now, although numerous obstacles were encountered at first, as is usually the case when opening up a new field.

Retorts for Zinc Distillation.—Circular retorts, used exclusively in the West, vary but little in dimensions at the different works. They are usually about 10.5 in. in external and 8 in. in internal diameter and from 48 to 54 in. long. Formerly they were made by hand, but now they are almost exclusively made by machine, either the ordinary auger-machine or a hydraulic press. The latter type of machine is coming more into use and no doubt is the better of the two; it makes a very dense retort, which prevents leakage of zinc-vapor through its walls. A pressure

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TABLE I.—*Data Relating to the Growth of the Zinc-Industry in the Middle West.*

Year of Construction.	Name of Company.	Abandoned.		In Operation.	
		No. of Retorts.	Total.	No. of Retorts.	Total.
	COAL-BURNING SMELTERS—ILLINOIS.				
1863	**Matthiessen & Hegeler Zinc Co., LaSalle**	.....	.....	3,344	.....
1869	**Illinois Zinc Co., Peru**	.....	.....	3,350	.....
1875	**Lumaghi Zinc Works, Collinsville**	244	.....	.....	.....
1886	**Collinsville Zinc Co., Collinsville**	.....	.....	1,536	.....
1892	**Waukegan Zinc Works, Waukegan**	.....	.....	672	.....
1891	**Winona Zinc Works, Winona**	.....	.....	448	.....
1897	**Sandoval Zinc Works, Sandoval**	.....	244	672	10,022
	MISSOURI.				
1869	**Missouri Zinc Works, Carondelet**	1,020	.....	.....	.....
1872	Martindale Zinc Works				
	Edgar Zinc Co. Carondelet.	.....	.....	2,000	.....
1873	Carondelet Zinc Works				
1881	**Joplin Zinc Works, Joplin**	1,120	.....	.....	.....
1882	**Rich Hill Zinc Works, Rich Hill**	.....	.....	672	.....
1887	**Robert Lanyon, Nevada**	1,344	.....	.....	.....
1896	**Geo. E. Nicholson, Nevada**	.....	3,484	672	3,344
	KANSAS.				
1874	**Weir City Zinc Works, Weir City**	1,200	.....	.....	.....
1878	**Robert Lanyon, Pittsburg**	1,200	.....	.....	.....
1881	**S. H. Lanyon & Bro., Pittsburg**	600	.....	.....	.....
1882	**W. & J. Lanyon, Pittsburg**	840	.....	.....	.....
1882	**Granby Mining & Smelting Co., Pittsburg**	912	.....	.....	.....
1888	**Scammonville Zinc Works, Scammonville**	400	.....	.....	.....
1886	**Girard Zinc Works (old), Girard**	448	.....	.....	.....
1890	**Girard Zinc Works (new), Girard**	896	.....	.....	.....
1891	**Pittsburg & St. Louis, Pittsburg**	.....	.....	896	.....
1890	**Weir City North Works, Pittsburg**	.....	.....	1,344	.....
1895	**Chicopee Zinc Co., Cherokee**	448	.....	.....	.....
1894	**Midland Smelting Co., Bruce**	1,120	8,064	.....	2,240
	Totals	.....	11,792	.....	15,606
	NATURAL-GAS BURNING SMELTERS—INDIANA.				
1892	**Columbia Zinc Works, Marion**	800	.....	.....	.....
1893	**Ingalls Zinc Works, Ingalls**	744	.....	.....	.....
1893	**Upland Zinc Works, Upland**	600	2,144	.....	.....
	KANSAS.				
1897	**Lanyon Zinc Co.** Robert Lanyon Sons Spelter Co., Iola.	.....	.....	.....	.....
1898	**Lanyon Zinc Co.** Robert Lanyon Sons Spelter Co., La Harpe.	.....	.....	9,600	.....
1897	**Lanyon Zinc Co.** W. & J. Lanyon, Iola.	.....	.....	.....	.....
1898	**Prime Western Zinc Co.** Prime Western Z Co., Gas City.	.....	.....	.....	.....
1902	**Prime Western Zinc Co.** Nicholson Z. Wks., Iola.	.....	.....	8,900	.....
1902	**Prime Western Zinc Co.** A. B. Cockerill, Gas City.	.....	.....	.....	.....
1898	**Cherokee-Lanyon Zinc Co., Gas City**	.....	.....	2,400	.....
1898	**Edgar Zinc Co., Cherryvale**	.....	.....	4,800	.....
1901	**United Zinc & Chemical, Iola**	.....	.....	2,400	.....
1902	**Granby Mining & S. Co., Neodesha**	.....	.....	2,400	.....
1903	**La Harpe Zinc Co., La Harpe**	.....	.....	2,100	.....
1903	**Chanute Zinc Co., Chanute**	.....	.....	1,280	.....
1904	**A. B. Cockerill, Altoona**	.....	.....	1,800	.....
1904	**Wm. Lanyon, Caney**	.....	.....	1,800	37,480
	Total in gas-district	.....	2,144	.....	38,080
	Total in coal-district	.....	11,792	.....	15,606
	Grand total	.....	13,936	.....	53,086

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of about 1,000 lb. per sq. in. is used. The capacity of both machines is about equal, but the cost of the hydraulic press plant is ten times that of the auger-machine.

Following the introduction of large furnaces, the method of operating has changed and numerous improvements have been made. Residues are now blown out from the retorts by a stream of water, instead of being scraped out with iron scrapers, as was formerly the practice.

Charging the Ore.—The ore-mixture is prepared outside of the furnace-building, instead of in front of the furnaces as formerly, and the charging is done in tiers instead of rows. The ore and coal are mixed mechanically and the mixture is placed in hopper-shaped charging-cars, from which it is charged into the furnaces. This mixture is usually prepared a day in advance, in order that it shall be ready for use when charging begins.

Drawing the Metal.—The metal is now drawn in large ladles holding from 300 to 400 lb., instead of the former smaller ones holding from 40 to 50 lb. These large ladles are suspended from cranes attached to trucks running on rails in front of the furnace and supplied with protecting-shields. In some cases the ladle is hung to a shield suspended from a single overhead-rail fastened to the furnace-front and traveling the whole length of the furnace.

At present the three sheet-zinc mills in the United States are in the western district. Matthiessen & Hegeler first rolled sheet-zinc in the West, followed by the Illinois Zinc Co.; both having sufficient capacity to roll their entire product of spelter into sheet. In 1902 the Lanyon Zinc Co. built a small mill at La Harpe, Kas., which has since been in constant operation.

Manufacture of Zinc Oxide.

While the manufacture of zinc oxide is confined principally to the eastern district, a few establishments are in the West. Generally the requirements for making zinc oxide cannot well be met in the West. In the first place anthracite coal, which is scarce in the West, is necessary, and cannot be had except at prohibitory prices. Oxidized zinc-ores free from cadmium are also necessary. Ores of this character occur in Wisconsin and the large plant of the Mineral Point Zinc Co., a branch of the New Jersey Zinc Co., has been erected at Mineral Point to

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treat these ores. A small plant, started in 1902 at West Plains, Mo., was soon absorbed by the New Jersey Zinc Co., and I believe is now out of commission. A third plant, the Ozark Zinc Oxide Co., built some years ago at Joplin, has so far remained independent, but difficulty was experienced at times in getting proper ore-supplies. At present these are obtained from New Mexico.

The zinc oxide industry will, in all likelihood, be controlled for some time to come by the New Jersey Zinc Co., which has practically an unlimited supply of ore admirably adapted to this purpose and abundant anthracite coal near its plants.

NOTE BY THE SECRETARY.—Comments or criticisms upon all papers, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

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The Fire-Clays of Missouri.

BY H. A. WHEELER, ST. LOUIS, MO.

(Lake Superior Meeting, September, 1904.)*

The Clay Industry of the United States.

It may surprise some of our members to learn, among the industries based on the mineral resources of the United States, that of clay now ranks third, being exceeded in value of product only by pig-iron and coal. The present annual value of clay-products exceeds the value of the outputs of gold and silver, and also exceeds the combined values of copper, lead, zinc, aluminum, mercury, antimony and nickel. These facts are shown in Table I. which gives the statistics for the year 1902:

TABLE I.—*Production of Metals, Coal and Clay in the United States During* 1902.[1]

	Value.
Pig-iron,	$372,775,000
Coal (including anthracite), $76,173,586,	367,032,069
Clay products,	122,169,531
Gold,	80,000,000
Copper,	76,568,954
Petroleum,	71,178,910
Silver (coin-value),	71,757,575
Lead,	22,140,000
Zinc,	14,625,596
Aluminum,	2,284,590
Mercury,	1,467,848
Antimony,	634,506
Nickel,	2,701

NOTE.—The value of the clay-products in the United States in 1903 was $130,962,648.

Of the total amount, $122,169,531 assigned for the value of the clay-products of the United States during 1902, about 20

* Presented at the Supplementary Session of the Institute in St. Louis.

[1] *Mineral Resources of the United States for* 1902, United States Geological Survey, Washington.

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per cent. is classified as pottery, or high-grade goods, and 80 per cent. as brick and tile, or low-grade ware. The latter is again subdivided into building brick, vitrified brick, fire-brick, sewer-pipe, terra-cotta, tiles and tiling, etc. The total value of the fire-brick produced in 1902 was $12,601,435, or about 10 per cent. of the value of the entire clay-products. Seven States furnished nearly 90 per cent. of the total quantity of fire-brick, which includes also stove-linings, fire-tiling, retorts, tuyeres, and other shapes produced by fire-brick manufacturers.

The Production of Fire-Brick.

The comparative value of the fire-brick industry in 1902 is given in Table II.

Table II.—*Production of Fire-Brick During* 1902.

Order of Importance.	Name of State.	Value.
1.	Pennsylvania,	$6,196,868
2.	Ohio,	1,520,442
3.	New Jersey,	828,057
4.	Missouri,	750,000
5.	Colorado,	609,495
6.	Kentucky,	605,448
7.	New York,	534,838
8.	Maryland,	298,840
9.	Alabama,	222,660
10.	Illinois,	199,048
11.	Montana,	113,112
12.	Other States,	702,628
	Total,	$12,601,435

The State of Pennsylvania ranks pre-eminently first in importance in the refractory-brick industry, and it is always likely to retain this position, as it is very rich in an extensive area of high-grade fire-clay in the central part of the State, and another large area of nearly as high-grade fire-clay in the southwestern portion, both of which occur at the base of the Coal Measures. These natural advantages, coupled with the best of cheap fuels, an enormous home-market in the iron-, steel- and coke-districts of Pittsburg and the Lehigh valley, and very low freight-rates, give it an unapproachable advantage over all other States. In this connection it is well to note that the consumption of fuel for making, drying and burning fire-brick is a

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large factor of their cost. Under the stimulus of the recent enormous expansion of the iron- and steel-business in the United States, the value of the fire-brick output in Pennsylvania has grown from $3,000,000 in 1898 to $6,117,000 in 1902.

Ohio, ranking second in the value of clay products, possesses an abundance of fair-grade fire-clays, has excellent cheap fuel, low freight-rates, and a large home-market in the iron- and steel-plants scattered throughout the State from Cleveland to the Ohio river; to a much less favorable degree, however, than Pennsylvania. The plants in southern Ohio adjoin the superior flint fire-clays of Kentucky, which promise to stimulate the production of the numerous fire-brick factories along the Ohio river.

New Jersey ranks third in importance, and although the oldest of the fire-brick producers, it is being slowly but surely outstripped by Missouri, Colorado and Kentucky. New Jersey has a variety of clays of Cretaceous age, many of which are refractory. Although the clay-producing district is limited in extent, the beds are usually thick and the quantity still available is very large. The clay-beds are at tidewater, and very low freight-rates are thus assured on clay- and fire-brick shipments to most of the large territory extending from Boston to Baltimore. Fuel, however, is somewhat high, when compared with Pennsylvania or Ohio, while the home-market, though large and varied, has probably reached if not passed its zenith, and will always be menaced by foreign importation.

New York, ranking seventh in the order of importance of clay, is merely the extension of the New Jersey clay-field, and therefore labors under the same disadvantages possessed by New Jersey.

Missouri ranks fourth, and is growing rapidly in spite of having its home-market heavily curtailed by the exhaustion of its important iron-ores. Twenty-five years ago there were 20 blast-furnaces, a large Bessemer plant, and several iron-mills in and about St. Louis, but the exhaustion of its once world-renowned beds of iron-ore at Pilot Knob and Iron Mountain has reduced this local iron- and steel-industry to two active blast-furnaces. While this staggering blow discouraged the development of new fire-brick plants, the old plants have been improved and enlarged. By developing the higher grades of ware, they are extending their goods to greater and wider

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horizons, and building up a broad business that is at least more healthy and reliable than when dependent upon a local specialized market. According to the statistics of the United States Geological Survey, the value of the output of clay-products in Missouri has grown from $290,000 in 1898 to $925,915 in 1903, though local returns indicate a much larger growth than this.

Colorado has recently pushed rapidly ahead to fifth place, under the stimulus of the rapid growth in the Pueblo iron-, steel- and coking-plants, and the concentration of the lead- and copper-smelting industries to the Rocky Mountain districts. Superior fire-clays of Cretaceous age occur along the eastern base of the mountains, and while fuel is inferior and high, the local industry is protected by the heavy freight-rates incident to a 1,000-mile up-grade haul from St. Louis, its nearest competitor.

Kentucky, now sixth in importance in the value of the annual output of clay-products, is growing rapidly. This State possesses at the base of the Coal Measures a very high-grade fire-clay that covers a large area, and with which is associated cheap, fairly good coal. The home-market, though small, is growing, and goods of high quality can be produced that will bear considerable transportation, especially as the railroads are disposed to be liberal in their freight-rates.

Maryland is of more importance historically than from its present or prospective output, although it possesses very superior fire-clay at the base of the Coal Measures in the Mount Savage district. The value of the output in 1902 amounted approximately to $300,000, and the industry is slowly diminishing, although the famous Mount Savage fire-brick has been on the market for more than 60 years. The competition of districts nearer to the iron- and steel-centers has interfered with the growth of the refractory-brick industry in Maryland, which is consequently limited to special classes of products.

The Missouri Fire-Clays.

The Missouri fire-clays are divided into two classes—the flint fire-clays and the plastic fire-clays—each having very different properties and geological occurrence. Both are used by Missouri fire-brick manufacturers, and each has special merits when

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properly used. The flint fire-clays are exceptionally pure, are devoid of plasticity, and if their excessive fire-shrinkage is removed by a preliminary burning, they make the most refractory or infusible fire-brick that it is possible to make out of clay, at least a brick that cannot be fused or injured by the highest heat attainable in a furnace fired by coal or coke. Although these bricks are ideal for withstanding heat, they are very unsatisfactory in shaft furnaces or under conditions involving abrasive action. Their tensile-strength or cohesion is so slight that fire-brick made exclusively of flint-clay rapidly wear out from the abrasive action of the descending-charge. In the arch of an open-hearth furnace, however, or in the checker-work of a regenerator, where only a "dry" or gaseous heat has to be withstood, they are very durable and satisfactory, notwithstanding the very high temperature attained.

The plastic fire-clays of Missouri are tough and plastic, and are only fairly fire-resistant on account of impurities present. These clays make a strong, durable brick that admirably withstands mechanical wear or abrasion, and are excellent for making gas- and zinc-retorts, linings for shaft-furnaces, lime-kilns, etc., but they cannot resist extremely high heats without "letting down" or softening, or fusing.

By skilfully combining the flint- and the plastic fire-clay in proper proportions, the experienced fire-brick maker can produce a brick that possesses ample refractoriness combined with a maximum strength or toughness, providing the customer clearly defines the conditions under which the brick is to be used. For most purposes the best fire-brick is composed of more or less pre-burned flint-clay, or "grog," the body or bond consisting of plastic clay. Too frequently the metallurgist considers only the refractoriness, or the purity as shown by a chemical analysis of a fire-brick and overlooks the mechanical weakness caused by the use of an excessive quantity of the non-plastic flint-clay; as a result, the fire-bricks are rapidly cut out by abrasion. Although not all of the fire-bricks makers are equipped with modern testing-laboratories using Seger-cones or Le Chatelier pyrometers, in most cases the manufacturer is better qualified than the metallurgist to design a fire-brick to meet given conditions; and longer campaigns and fewer re-linings of furnaces would result if this responsibility were

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transferred to intelligent, up-to-date manufacturers. Some of the Missouri fire-brick makers do not belong to that class, but continue to claim, as did their revered grandfathers, a heat-resistance of from 4,000° to 5,000° F. for their products, although the Le Chatelier pyrometer shows them to be unsafe at 2,600° F.

The St. Louis Fire-Brick Market.—The St. Louis market furnishes ten well-defined grades of fire-brick, ranging in heat-resistance from 2,300° to 3,000° F., and in value from $14 to $35 per 1,000 f.o.b. cars at factory. These fire-brick are manufactured by the old hand-press or slop process, and by brick machines of the auger and semi-dry types. With regard to the respective merits of hand-made vs. machine-made brick, the old hand-press unquestionably makes a more refractory and superior brick, but the machine-made brick is more uniform and economical, and is now made with such care as to be quite satisfactory. The machine-made brick is also becoming of much greater importance in these days of labor troubles, by greatly reducing the quantity and skill of labor required for a large output.

The marked range in refractoriness and price of fire-brick gives no clue whatever of its value to the metallurgist, since the cheapest brick, which is made entirely of plastic clay, would prove far more durable in the hearths of roasting-furnaces, linings for water-jacketted furnaces and lining of lime-kilns, where abrasive action is severe, than those of higher price; but the heat-resistant quality of the cheaper brick ranges only from 2,900° to 2,200° F., while that of the highest-priced flint fire-brick is capable of withstanding a temperature of 3,000° F.

Occurrence of Flint Fire-Clays.—The flint fire-clays of Missouri occur in the eastern-central portion of the State, at a distance of from 40 to 140 miles west of St. Louis, along the Wabash, Colorado or Rock Island, Missouri-Pacific and Frisco railroads. The clays are found in local pockets of very variable size that are possibly of Cretaceous age in limestones of Sub-Carboniferous and Lower Silurian age. The pockets are evidently old sink-holes in the limestone that have subsequently become filled with clay, and probably at such an extremely slow rate as to permit of the almost complete removal of the impurities by chemical solution, including some of the silica of the kaolinite

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base. The pockets are fringed with a sandy transition-bed that terminates in a sandstone which always flanks the sink-hole. This sandstone represents the silt or coarse matter that was promptly deposited, as a delta, by the inflowing streamlets that brought the detritus from the adjacent weathering limestone in rainy weather, while the finer, clayey matter was deposited in the central portion of the sink-hole. The sandstone always dips on all sides towards the center of the pocket, at angles of from 15° to 35°. The laminations of the clay-deposit have been destroyed by the chemical action of the water that filled the basin, so that the flint-clay has a very characteristic hard, massive, jointed structure, and breaks with a sharp conchoidal or flint-like fracture that gives the clay its name.

Most of the pockets are filled with a light-gray, exceptionally pure flint-clay that usually contains from 1 to 2.5 per cent. of impurities; occasionally it is more or less heavily contaminated with red iron oxide, which of course, ruins it for refractory purposes. The flint-clay has a hardness of from 3 to 3.5 and is devoid of plasticity, unless very finely ground, even then it is only slightly plastic. Even the weathered outcrop of flint-clay is usually devoid of plasticity and consists of small, sharp, angular fragments.

The pockets resemble most sink-holes in being usually circular in shape and of very variable size, ranging from 25 to 300 ft. in diameter and from 10 to 40 ft. in depth. A sink-hole may produce from 5 to 500 car-loads before becoming exhausted. The flint-clay is usually worked on leases by local shippers, who pay the farm-owners a royalty of about 5 cents per ton. The pockets are worked in open pits and, with a haul of from 0.5 to 3 miles over poor roads, the flint-clay usually sells for $1 per ton f.o.b. cars at local points, or from $1.50 to $2 per ton at St. Louis.

Chemical Properties of Flint Fire-Clays.—The chemical analysis of the Missouri flint-clays is especially interesting not only for being very low in impurities, but in having from 0.5 to 3.5 per cent. excess of alumina above the amount contained in pure kaolinite. According to Dana and other leading authorities, kaolinite, the base of all clays, contains: Silica, 46.3; alumina, 39.8; combined water, 13.9; and total, 100 per cent.

The flint-clays of Missouri vary in these three essential con-

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stituents, as follows: Silica, 40.8 to 48.; alumina, 37. to 43.2; combined water, 13.3 to 15. per cent.

The water also is apt to be abnormally high as compared with pure kaolinite.

The percentage of fluxing-impurities—iron oxide, lime, magnesia and the alkalies—ranged in nine average samples collected from the various shipping-points, as follows: Iron sesquioxide, 0.15 to 0.83; lime, 0.28 to 1.93; magnesia, trace to 0.58; alkalies, 0.2 to 1.2; total, 1. to 2.89 per cent.

Such remarkable freedom from fluxing-impurities is very exceptional, and as these analyses are based on average samples collected over the face of the entire clay bank, they must not be confused with the misleading analyses of selected specimens that are frequently found in text-books and in general technical literature. The influence of the adjacent source of supply of the clays, or the limestone, is noticeable in forming the largest portion of the impurities.

This great purity, combined with a coarse grain and high density, the latter being also very important factors of refractoriness,[2] is the reason why these flint-clays are the most refractory of all known clays and are able to withstand a temperature of 3,000° F. with safety.

The flint-clays not being plastic, it is necessary to mix them with a plastic or "bond" clay in order to mold them into bricks, and local St. Louis fire-clay is added, either raw or after washing, for the St. Louis market. Efforts made to bind the flint-clays with very finely ground flint-clay have met with sufficient success to produce a brick, but it is tender and quite incapable of withstanding abrasion or careless handling in transit.

On account of the large amount of water contained in flint-clays that is expelled at a red heat, the shrinkage is excessive on burning, so it is necessary to calcine the clay before using it in order to eliminate the heavy fire-shrinkage, which ranges from 10 to 14 per cent.

Occurrence of Plastic Fire-Clay.—The plastic fire-clays of Missouri occur at the base of the Coal Measures in the eastern portion of the State in two different basins, known respectively as the St. Louis and Mexico areas. The St. Louis basin is the ex-

[2] Clay-Deposits of Missouri, by H. A. Wheeler, *Missouri Geological Survey*, vol. xi., p. 133.

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treme western fringe of the Illinois coal basin, which overlaps into St. Louis and St. Charles counties in Missouri. The Mexico basin is the eastern fringe of the Coal Measures that cover central western Missouri, and which is quite distinct from the Illinois Coal Measures.

Occurrence of St. Louis Fire-Clay.—The St. Louis basin, though much smaller in size, is much more important than the Mexico basin. It covers the western portion of the city of St. Louis, a large part of St. Louis county, and a small part of St. Charles county. The entire area, including a few small detached outlying-basins, is in the form of an irregular polygon, and aggregates about 170 sq. miles. While there are several sheets of clays and shales in the lower Coal Measures in this basin, only one, the St. Louis fire-clay seam, is refractory, or capable of withstanding a heat of 2,500° F. which I regard as the line of demarcation between the refractory and non-refractory clays (this temperature being that of a pure white heat). This seam, which rests on the ferruginous sandstone, or the basal member of the Coal Measures, is a sheet of typical Coal Measure clay varying from 4 to 40 ft. in thickness, usually from 6 to 8 ft. It is nearly horizontal, has a roof of from 1 to 22 in. of bituminous coal, and rests on a transition-bed of sandy, green clay that merges into the ferruginous sandstone.

The fire-clay outcrops at the base of the valleys, where it is entered by adits, but on the higher ground it is worked by shafts from 25 to 125 ft. deep.

The St. Louis fire-clay has a light- to dark-gray color, and is a hard, rock-like clay when freshly mined that rapidly disintegrates into a soft, tough, plastic paste on exposure to the weather or when ground with water. It is worked on the "room and single-entry" system, and mined by hand-picking a side-cut along the pillar, and then blasting to this cut with black powder. Formerly this work was done by contract at from 35 to 50 cents per cubic yard (equivalent to about 2 tons), but it is now paid for by day-work, the miners receiving about $2 per day. While most of the fire-brick plants own their own clay-mines, there are a number of independent leasers operating on a royalty basis of from 3 to 5 cents a ton, who sell their output to St. Louis and to distant consumers. The principal mines are in the western part of St. Louis in what is known as the

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Cheltenham district, and the St. Louis fire-clay sells for from 60 cents to $1 a ton f.o.b. cars for mine-run, and from $1.50 to $2 for selected clay. These prices are about one-half the prices of New Jersey fire-clays.

There is a specially selected grade known as "pot-clay," used for making glass-pots, which is obtained from a purer, more uniform seam that occasionally occurs from 1 to 4 ft. thick in the middle or upper part of the bed. This pot-clay sells for from $10 to $14 a ton f.o.b. cars St. Louis.

The St. Louis fire-clay bed varies considerably in thickness, character and value. While the general character of the clay is coarse-grained and moderately siliceous, the silica ranging from 58 to 63 per cent., it occasionally becomes very coarse and sandy, and the silica increases to 70 per cent.; more frequently it becomes finer-grained, and the silica decreases to 50 per cent. The clay always contains more or less iron pyrite ("shiners," "sulphur" or "mundic"), which is usually disseminated throughout the clay in the form of fine crystals to the extent of from 1 to 3 per cent., but at times it segregates into coarse granules and nodules. The latter form is exceptional, and while very easy to remove by washing, if allowed to get into the fire-brick it causes local fusion and a black slag. The redeeming feature of the excessively high percentage of iron in the St. Louis fire-clay lies in the fact that it is nearly always finely and uniformly disseminated through the clay—in fact, the pyrite crystals are usually too small to be removed by washing. It is the iron oxide that gives the brown color to fire-brick made from St. Louis clay when hard burned, while the fine pyrite crystals produce the small black specks. If underburned, the fire-bricks are from light-colored to white.

When the fire-clay occurs abnormally thick, as in the northern part of St. Louis county, the lower portions of the bed are more or less contaminated by calcareous concretions that resemble boulders of limestone; they ruin the clay for refractory uses, unless removed by careful sorting and washing.

Physical Properties of the St. Louis Fire-Clay.—The St. Louis fire-clay, after weathering or grinding, is more or less plastic. As measured by the tensile-strength test, it ranges from 80 to 150 lb. per sq. in. in air-dried briquettes, and the prolonged weathering of several years develops as high as 250 lb. A

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fire-clay of average plasticity shows a tensile-strength of about 100 lb. per square inch.

When made up into a stiff mud with from 14 to 20 per cent. of water, the clay shrinks from 6 to 9 per cent. in air-drying; the air-dried clay shrinks from 4 to 8.5 per cent. when burned to vitrification, which is the limiting-point of shrinkage; and the total shrinkage varies from 11 to 16 per cent. The clay can be rapidly dried without cracking, and more or less rapidly heated without checking.

Incipient vitrification begins at from 2,100° to 2,250° F.; complete vitrification, or the safety-limit, occurs at from 2,300° to 2,450° F.; and viscosity, or initial fusion, takes place at from 2,500° to 2,700° F.; hence the clay is only moderately refractory.

The hard-burned ware forms a close body of great strength, which largely contributes to its very high value for zinc-retorts and glass-pots.

The St. Louis fire-clay is coarse-grained and has a high specific gravity, ranging from 2.40 to 2.47 in the mine-run, and from 1.92 to 2.13 in the washed clay. It is largely this coarse grain and high density that renders the St. Louis fire-clay as refractory as it is, with its unusually high percentage of fluxing-impurities (for a fire-clay). In fact, extensive work on the Missouri clays, using the Le Chatelier pyrometer for the heat-measurements, has clearly brought out the fact that it is impossible to determine even approximately the refractoriness of a clay from the chemical analysis, because density and size of grain are found to have such important influence on the fusibility.

Chemical Properties of the St. Louis Fire-Clays.—The St. Louis fire-clay is most interesting in its chemical composition, for the reason that its high percentage of fluxing-impurities, combined with its high silica, would cause it to be unqualifiedly condemned for a refractory clay by the analytical chemist. As a matter of fact, it is unequaled in the United States for its superiority in making glass-pots, zinc-retorts and gas-retorts, while it makes an unusually durable fire-brick when the heat is not excessive. These statements allude to ware made exclusively ot St. Louis fire-clay, without the addition of Missouri flint-clay to improve its refractoriness.

The fluxing-impurities—iron oxide, lime, magnesia and alkalies—usually range from 5 to 6 per cent., seldom as low as 4.5

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per cent. and occasionally as high as 7.3 per cent. Good fire-clays are usually stated in published analysis to range from 2 to 4 per cent. of impurities, and as a 1-per-cent. increase in the quantity of impurities present frequently lowers the fusibility at least 100° F., the chemist would be disposed to condemn all the St. Louis fire-clays for refractory requirements of even moderate demands. A high content of silica, which is always due to more or less free silica when it exceeds 46 per cent., also adds to the fusibility of a clay to a small but undetermined extent. Formerly it was supposed to be a harmless diluent, like titanic acid, but the researches of Seger seem to leave no doubt of the detrimental action of free silica in a refractory clay, though probably not to the extent he attributes.

In considering the preceding and subsequent analytical results, it is important to remember that the Missouri analyses were made on samples that were carefully taken over the entire clay-face or dump by disinterested experts, and are therefore actual averages of working-samples, whereas many quoted analysis are on selected specimens that have too frequently been taken by interested parties.

The data given in Table III., based on air-dried average samples, gives the variations and average composition of the St. Louis fire-clay as obtained from seven different mines and four different washing-plants.

TABLE III.—*Composition of St. Louis Fire-Clays.*

	Mine-Run.		Washed.	
	Per Cent.	Average Per Cent.	Per Cent.	Average Per Cent.
Combined silica (SiO_2)	24.6 to 34.4	32.0	26.0 to 35.5	32.0
Free silica (SiO_2)	19.5 to 42.9	30.0	20.7 to 37.3	25.0
Alumina (Al_2O_3)	19.3 to 30.8	24.0	21.1 to 28.8	24.0
Combined water (H_2O)	7.7 to 11.9	10.5	8.9 to 11.4	10.0
Titonic oxide	trace to 2.0	1.0	trace to 1.9	1.1
Iron sesquioxide (Fe_2O_3)	0.8 to 4.7	1.9	1.8 to 2.5	1.85
Iron protoxide (FeO)	0.4 to 1.8	1.2	0.8 to 1.2	1.0
Lime (CaO)	0.4 to 1.2	0.7	0.6 to 0.9	0.7
Magnesia (MgO)	0.1 to 0.9	0.3	0.07 to 0.8	0.2
Potash (K_2O)	0.2 to 0.9	0.5	0.5 to 0.6	0.55
Soda (Na_2O)	0.2 to 0.6	0.2	0.1 to 0.4	0.10
Sulphur (S)	0.1 to 0.7	0.3	0.1 to 0.3	0.18
Sulphuric acid (SO_3)	0.1 to 0.6	0.35	0.2 to 0.6	0.4
Moisture	2.0 to 4.1	2.7	3.0 to 4.5	3.0
Fluxing impurities	4.3 to 7.3	5.5	4.7 to 5.2	4.8
Specific gravity	2.40 to 2.47	2.44	1.92 to 2.13	2.0

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The St. Louis Fire-Brick Industry.

There are seven plants in St. Louis, and one in the adjoining St. Louis county, engaged in the manufacture of fire-brick and the usual tiles and special shapes called for by the refractory trade. Three make also gas-retorts that are shipped throughout the United States from the Atlantic to the Pacific coast. The plant in St. Louis county is only 5 years old, but the others are from 20 to 50 years old.

About $2,500,000 are invested in these plants and the annual value of the output in 1903 approximated $1,400,000, which is exclusive of a large output of sewer-pipe produced by two of the larger plants. These figures, while considerably larger than those given in the clay statistics published annually by the U. S. Geological Survey, are regarded as more reliable; and, in addition, from $125,000 to $150,000 worth of fire-clay is produced annually by independent leasers, most of which is shipped to nearly all of the zinc-smelters in the United States, who depend on the St. Louis fire-clay for their zinc-retorts.

The Mexico, Mo., Fire-Clay.

The extent of the Mexico fire-clay basin has not yet been defined, and the workable fire-clay has only been found along the fringe, or at or near the outcrop, of the basal Coal Measures. It contains only one bed of refractory clay which has been worked at Fulton, Mexico and Vandalia, or over a distance of 50 miles, while outliers are known to occur beyond these limits. The deposit may be said to be an irregular parallelogram at least 75 miles long by from 5 to 15 miles wide.

The fire-clay rests on the ferruginous sandstone, with an intermediate transition-bed of sandy clay more or less stained green by iron. The seam is usually thick, varying from 6 to 40 ft., but only the lower 6 to 12 ft. of it is sufficiently pure for use as a refractory material. It is mined through shafts ranging from 40 to 150 ft. deep.

The clay is light- to dark-gray, hard and rock-like, and is blasted by the room-and-entry system. Considerable timbering, however, is required to support the roof.

Physical Properties.—The clay, after weathering or grinding, makes a fairly plastic paste that has a tensile strength of from 40 to 80 lb. per sq. in. when dried. When molded with from

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14 to 16 per cent. of water, it has a shrinkage of from 4 to 5 per cent., and an additional shrinkage when burnt to vitrification of from 6 to 7 per cent., giving a total shrinkage of from 10 to 12 per cent.

Incipient vitrification begins at from 2,200° to 2,300° F., complete vitrification, or the danger line, occurs at from 2,400° to 2,500° F., while viscosity, or initial fusion, takes place at from 2,600° to 2,700° F.

Chemical Properties.—The fire-clay of the Mexico basin is much purer than that of the St. Louis seam, the proportion of iron oxide, especially, being much lower. The total percentage of fluxes range between the usual limits of most refractory clays.

The variation in composition is shown in Table IV. which is based on analyses of samples from the four principal mines in the Mexico district:

TABLE IV.—*Analyses of Clays of the Mexico District.*

	Per Cent.	Average Per Cent.
Silica,	47.0 to 55.0	52.0
Alumina,	30.0 to 37.0	33.0
Combined water,	10.5 to 13.0	12.0
Iron sesquioxide,	1.2 to 1.8	1.5
Lime,	0.4 to 0.7	0.5
Magnesia,	0.0 to 0.3	0.7
Alkalies,	0.5 to 1.4	0.7
Total fluxes,	2.5 to 4.2	3.4
Specific gravity,	2.43 to 2.46	2.44

THE FIRE-BRICK INDUSTRY OF THE MEXICO BASIN.

A plant of moderate size has been in operation at Vandalia and another at Fulton for about 20 years, and a third plant was operated at Mexico until 1895. In 1896 a small plant was built amid the flint-clays at Truesdale, but it had a short life. These plants have the disadvantage of no freight-competition; their coal costs considerably more than at St. Louis, and there is no home-market even for boiler-grade products. It is not surprising that this district has shown no growth in recent years in competition with the more favored St. Louis district. The capital invested aggregates about $150,000, and the annual value of the output is about $100,000.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Decomposition and Formation of Zinc Sulphate by Heating and Roasting.

BY PROFESSOR H. O. HOFMAN, MASS. INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

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PART I. DECOMPOSITION OF ZINC SULPHATE BY HEATING IN AIR.

I. *Introduction.*

WITH the exception of lead sulphate, all common metallic sulphates are completely decomposed upon heating into metallic oxide, sulphur trioxide, sulphur dioxide and oxygen. Some give up their trioxide readily at a low temperature, others

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require considerable heat and much time to be completely freed from sulphur. Kerl[1] in 1881 arranged the principal metallic sulphates as they are decomposed by a rising temperature in the following order: silver, iron, copper, zinc, nickel, cobalt, manganese and lead; lead sulphate being decomposed only slightly at a white heat. The researches of Bradford,[2] published in 1903, show, however, that ferrous sulphate is decomposed at 590° C., cupric sulphate at 653° C. and argentic sulphate at 1,095° C. By the present investigation it will be seen that zinc sulphate is decomposed at 739° C. As far then as our present knowledge goes, the order in which the decomposition of the leading metallic sulphates takes place is the one given in Table I.

TABLE I.—*Temperatures of Decomposition of Various Metallic Sulphates.*

Fe	Cu	Zn	Ag_2	Ni	CO	Pb	SO_4
590° C.	653° C.	739° C.	1,095° C.	n. d.	n. d.	n. d.	°C.

Very little work has been done to determine the temperature at which zinc sulphate is decomposed, the rapidity with which the decomposition takes place, and the products that are formed. Schmieder[3] approximated the temperature of complete decomposition of zinc sulphate in a hand-reverberatory roasting-furnace treating blende by placing near the fire-bridge Seger cones and noting the one that melted when basic sulphate was converted into oxide. He estimated the temperature to be about 1,100° C.; that is, the hearth near the fire-bridge has to reach about 1,100° C., if the blende is to be satisfactorily dead-roasted. As to the time required for decomposing zinc sulphate, it is known in a general way that dead-roasting blende is a lengthy operation on account of the difficulty of decomposing the last of the sulphate. Eichhorn,[4] in connection with roasting blende in a muffle furnace, calls attention to the fact that it is not so much temperature as time that is required to

[1] *Grundriss der Metallhuettenkunde*, Leipsic, 1881, p. 70.
[2] *Trans.*, xxxiii., 50.
[3] *Œsterreichisches Jahrbuch*, 1889, p. 393.
[4] *Berg- und Huettenmœnnische Zeitung*, 1889, p. 113.

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obtain satisfactory results. In regard to the products formed, Vogel,[5] as early as 1814, brought normal zinc sulphate to a red heat and obtained a product insoluble in water which contained sulphur. He called it basic sulphate. Plattner[6] says that in roasting blende there is formed, beside the normal sulphate, a basic sulphate of definite chemical composition, the tetrabasic sulphate 3 ZnO, $ZnSO_4$, and that while the former is readily decomposed upon raising the temperature above the normal for roasting, into ZnO, SO_3, SO_2 and O, the basic salt requires a white heat for the complete elimination of sulphur. Plattner is the only investigator who gives a basic salt formed by heating normal sulphate a definite formula. The present investigation makes it probable that, in decomposing zinc sulphate by heat, the sulphur trioxide passes off gradually and that no definite basic compound is formed.

A series of basic salts crystallizing with various amounts of water has been isolated from solutions, but they have no bearing on the work in hand.

Some of the preliminary work in this investigation was done by Messrs. W. A. Dorey and R. P. Roberts in 1900. It was carried further in 1901–02 by Mr. A. C. Dart, Jr., who tried to prove or disprove, by heating zinc sulphate in a current of air and of carbon dioxide, the correctness of the generally accepted statement that heating a metallic sulphate with limited access of air caused the SO_3 to split to a large extent into SO_2 and O, while with free access of air the volatile compound driven off was mainly SO_3. His results are given in Table II.

TABLE II.—*Results of Heating Zinc Sulphate With and Without Excess of Air.*

Heated in	Temperature.	Total Sulphur Eliminated.	Sulphur Eliminated as SO_2.	Sulphur Eliminated as SO_3.	Ratio: $\frac{S \text{ as } SO_2}{S \text{ as } SO_3}$
	Degrees Centigrade.	Per Cent.	Per Cent.	Per Cent.	
Air	578	1.63	0.14	1.49	9.06
Air	584	1.79	0.14	1.65	8.26
Carbon dioxide	599	0.50	0.18	0.32	55.25
Carbon dioxide	602	0.50	0.16	0.34	47.88

[5] *Journal fuer Chemie und Physik* (edited by J. S. C. Schweigger), 1814, xi., p. 416.

[6] *Die Metallurgischen Roestprocesse*, Freiberg, 1856, pp. 87, 142.

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In the experiments, the results of which are given in Table II., the temperature of the furnace was brought slowly to the point at which the first acid was given off, raised about 10° C. and then maintained constant for several hours. While only a very small proportion of the total sulphur was driven off in the tests, they show that in heating zinc sulphate the tendency of the salt to split into ZnO, SO_3 and O is greater when oxygen is absent than when it has free access. Schnabel[7] quotes Schlapp as having found, while decomposing zinc sulphate on a large scale by heating, that of the total SO_3 present, 30 per cent. escaped as such, while the rest passed off as SO_2 and O. Investigations along this line were not further pursued, as the researches of Knietsch[8] on the formation and decomposition of sulphur trioxide by heat in the presence and absence of catalyzing agents had about covered the ground.

The present investigation was taken up in 1903. A grant by the American Academy of Arts and Sciences from the income of the C. M. Warren fund enabled me to avail myself of the assistance of Mr. R. B. Yerxa in carrying on the work.

II. *Zinc Sulphate Used.*

The zinc sulphate which formed the raw material for the experiments was the chemically pure normal salt, $ZnSO_4 + 7\,H_2O$. It was analyzed for zinc by the standard sodium carbonate method, and for sulphur by precipitating with barium chloride, the zinc having been first removed to prevent any of it from being carried down by the barium sulphate. The water was found by the difference. The results of the analysis are given in Table III.

TABLE III.—*Composition of Zinc Sulphate.*

	By Analysis.	By Calculation.
	Per Cent.	Per Cent.
SO_3	28.78	27.84
ZnO	29.27	28.32
	By difference.	
H_2O	41.95	43.84
Ratio : SO_3 : ZnO...	0.9833	0.9828

[7] *Handbuch der Metallhuettenkunde*, Springer, Berlin, 1904, 2d ed., vol. ii., p. 46.

[8] *Berichte der Deutschen Chemischen Gesellschaft*, 1901, xxxiv., 4,069; transl. by Ruhoff, *Mineral Industry*, 1901, x., 605.

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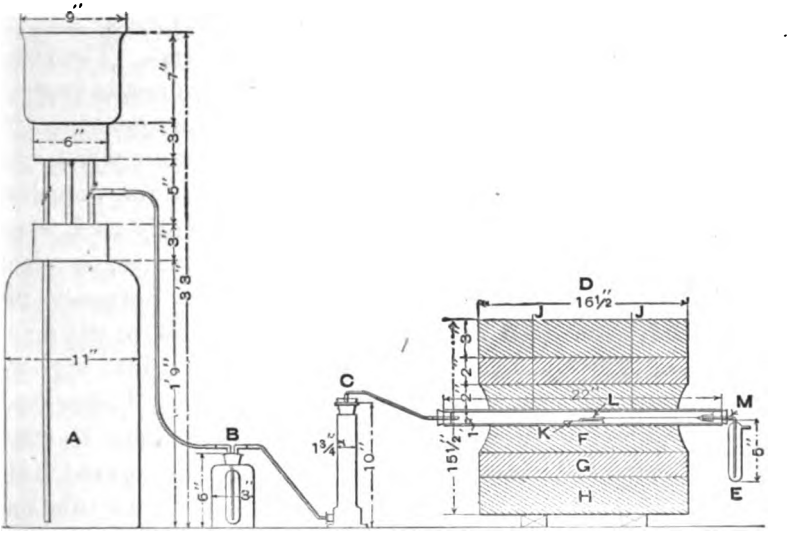
The results in Table III. show that the analysis gave about 2 per cent. less water than the quantity required by the formula. This difference was accounted for by the fact that the salt had been kept in a closet the temperature of which had reached sometimes 30° C. Upon close examination the crystals appeared dull, as if they had weathered. The close agreement of the two ratios of $SO_3 : ZnO$ proves that the sulphate was the normal salt.

FIG. 1.



ELECTRIC TUBE-FURNACE WITH GAS-TRAIN.

A, Gasometer; B, Ahlin wash-bottle with permanganate; C, Drying tower; D, Tube-furnace; E, Railroad tube with indicator-solution; F, Calcined magnesia; G, Asbestos-magnesia; H, Fluted asbestos steam-pipe covering; J, Porcelain tube; K, Platinum boat; L, Thermo-couple; M, Rubber stopper.

III. *Furnaces and Pyrometers.*

Two electric furnaces and one gas-furnace served for the heat-work. The electric furnaces were of the resistance type first suggested and used by Prof. C. L. Norton[9] in 1894, and since then commonly employed for laboratory heating-purposes with temperatures up to about 1,400° C. Fig. 1 represents a sketch of the tube-furnace, D, with gas-train, A, B, C. A porcelain tube, J, J, 22 in. long and 1 in. in internal diameter, glazed

[9] Fay-Badlam, *Technology Quarterly*, 1900, xiii., 304; Norton, *Iron Age*, Jan. 3, 1901, p. 22.

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inside only, was wound in the center for a distance of about 8 in. with 0.5-mm. platinum wire. The platinum coil, of 30 turns about 0.25 in. apart, was coated with a thin mortar of chemically pure magnesia; the tube then covered, excepting 5 in. at the ends, with a thick mortar of calcined magnesia, F; this surrounded with ordinary asbestos-magnesia covering, G; and the whole enclosed with fluted asbestos steam-pipe covering, H. The sides of the furnace were firmly held together with brass ribbons in common use in covering steam-pipes, and the ends plastered over with a mixture of magnesia and water-glass to prevent the dry part of the lining from falling out. The platinum coils were coated with chemically pure magnesia to prevent them from coming in contact with the impurities contained in the ordinary magnesia, as these often form clinkers which corrode the platinum wire. By leaving the ends of the porcelain tube uncovered for 5 in., they remained sufficiently cool, when the furnace was brought to the highest temperature, to admit the use of rubber stoppers. The wires of the thermo-couple were firmly connected with the rubber stopper, M, at the delivery-end of the tube by cutting a longitudinal slit into opposite sides, inserting a wire, and binding the whole with adhesive tape. By thus fastening the wires to the stopper, the couple, L, when once adjusted to be in contact with the substance in boat, K, to be heated, would be again in correct position when the tube was closed up after the stopper had been removed for some reason or other. The wires reaching into the tube were insulated from one another, yet, at the same time, firmly held together by winding them over and under, with asbestos string.[10]

The gas-train consists of a gasometer A of ordinary type used for forcing air or any other gas through the apparatus. It is followed by an Ahlin wash-bottle B, filled with a strong solution of potassium permanganate, which oxidizes all traces of organic matter or sulphurous acid from the air. The drying tower C, filled with pieces of calcium chloride, removes any moisture present, so that only thoroughly purified and dried air can enter the porcelain tube of the furnace. At the opposite end is a railroad tube, E, to hold the indicator solution. It was found necessary to have two indicators. One was a very dilute

[10] Le Chatelier-Boudouard-Burgess, *High Temperature Measurements*, Wiley, New York, 1901, p. 113.

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solution of iodine in potassium iodide and of starch to show the presence of sulphurous acid by the fading of the blue color. As the blue color is also destroyed by organic matter, another indicator had to be provided for carbon-reduction experiments. At the suggestion of Prof. A. H. Gill, a colorless solution of potassium iodate and starch was used which was changed to blue by sulphurous acid. Carbon burnt in the tube had no effect upon it.

The tube-furnace was adopted after experimenting for some time with a similar electric furnace, 16 in. long by 10 in. in diameter, the chamber of which consisted of a cylindrical clay retort, 6 in. long by 2.5 in. in diameter, the mouth being flush with the covering to admit access with tools. The furnace had to be given up for experiments requiring hermetic sealing, as it was found impossible to cement a plug into the mouth of the cylinder which would be air-tight during the experiment and permit of being easily removed at the close of a test. The retort-furnace was employed, however, in conjunction with the tube-furnace for dehydrating and for heating larger quantities of zinc sulphate than the tube could conveniently hold under conditions not requiring the furnace to be air-tight. A gas-muffle furnace, made by the American Gas Furnace Co., New York, N. Y., with muffle 12 in. long by 6.75 in. wide by 3.75 in. high, was used for heating large quantities of zinc sulphate and for roasting blende.

The temperatures were measured with Le Chatelier thermo-electric couples. With the tube- and retort-furnaces a Chauvin-Arnoux galvanometer was employed, with the gas muffle-furnace the common form of Deprez-d'Arsonval. The couples were calibrated against the boiling-points of water, naphthalene and sulphur, and the melting-points of aluminum and gold.

IV. *Heating of Anhydrous Zinc Sulphate—Dehydration.*

Manuals of chemistry teach that hydrous zinc sulphate loses six of its seven molecules of water when it is heated to 100° C.; that, at a higher temperature, the last molecule is driven off; and that it is difficult to remove this without at the same time starting the decomposition of the anhydrous salt. As it was important in the proposed work to start with anhydrous salt,

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tests were made to find the temperature at which zinc sulphate was completely dehydrated. Weighed samples of the crystalline salt were partly dehydrated on a steam-table and then transferred to the tube-furnace. The temperature of the steam-table was about 109° C. and caused an average loss in weight of 36.29 per cent. In the tube-furnace the temperature was first brought to and maintained at 238° C. to constant weight; it was next raised to 324° C. and again held until there was no further diminution in weight. Increasing the heat did not cause any additional loss until the sulphate began to be decomposed. The temperature of approximately complete dehydration was, however, lower. Further tests at 240°, 260° and 280° C. showed that above 280° C. there was no additional loss. Working within the limits of 260° and 280° C. it was found that 263° C. was the lowest temperature at which the last water (excepting 0.24 per cent.) was given off. With samples weighing 2 g. each, it took from 30 to 40 hours to finish a test. The average loss of water in the runs made at 263° C. or slightly above was 41.71 per cent., while the analysis of the salt calls for 41.95 per cent. The last 0.24 per cent. water could be set free only at a temperature at which the decomposition of the sulphate began. The losses in weight from 100° to 263° C. showed that the quantity of water driven off rose with the temperature, as did the rapidity with which it passed off.

V. *Heating of Anhydrous Zinc Sulphate—Decomposition.*

(*a.*) *Beginning of Decomposition.*—Five separate experiments were carried through to determine the beginning of decomposition of anhydrous zinc sulphate. The mode of operating was as follows: A sample of the hydrous salt weighing about 3 g. was partially dehydrated on a steam-table, transferred to the tube-furnace, heated to constant weight, set aside to free the cool end of the delivery-tube completely from condensed moisture, and then replaced in the tube. In the preliminary trials it has been found that this condensed moisture absorbed the first traces of acid set free and vitiated the results. With the sample again in place and the tube closed, the air-current was started and the furnace heated, rapidly to 500° C. and then slowly. Galvanometric readings were taken as a rule every minute, often, however, at shorter intervals. When, in the

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slow heating, the temperature was raised 5° C. a minute, the first fading of the indicator occurred at 550° C.; when more slowly, it took place at a lower temperature. The lowest point recorded for the beginning of decomposition on a slowly rising temperature was 532° C. Upon lowering the temperature from 532° to 525° C. and allowing it to remain constant, no fading of the indicator could be detected. When it was increased to 528° C. and held there, the indicator began to fade after about 10 minutes. The decomposition of anhydrous zinc sulphate begins at 532° C. with a rising, and at 528° C. with a constant, temperature. The difference of 4° C. finds its explanation in the time required for the extremely small quantity of acid set free to travel from the boat, holding the charge, to the indicator tube. The evolution of sulphurous gas at 528° C. lasted for from 1 to 2 hours, according to the weight and the thickness of the charge. In two tests in which the salt was weighed after the sulphurous gas had ceased to be evolved, the loss in weight was only 0.04 and 0.05 per cent., based on that of the anhydrous salt, which was about 2 g. It ought to be mentioned that, after one test was finished and before another was begun, the temperature was lowered well beneath the decomposition point; also, the furnace was swept clean from any remaining free acid by a current of air.

(*b.*) *Rapidity of Decomposition.*—In attempting to determine the rapidity of decomposition it was necessary to feel one's way before any systematic work could be undertaken. A sample corresponding to 1.7535 g. of anhydrous zinc sulphate was first heated to 528° C. and kept at this temperature for 1.5 hours, when acid ceased to come off. No further decomposition took place with increase of temperature until 575° C. was reached. It took 3.5 hours of heating to separate the acid. Raising the temperature again slowly, the indicator showed the presence of acid at 610° C., and 52.5 hours passed before acid ceased to come off. As it required at the end of the 52.5 hours one whole hour's heating for the indicator to show signs of fading, the temperature was raised more quickly, 10° C. per hour, until 681° C. was reached and maintained at this stage for 4 hours. A further rise of 9° C. brought the charge to 690° C., at which temperature it was again held for 4 hours. The result of the 76 hours' heating was a loss of only 7.5 per cent. of SO_3 out of

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a possible of 49.6 per cent. This decrease in weight was checked by a determination of sulphur in the residue, which was found to be 16.8 per cent. The 7.5 per cent. of SO_3 correspond to 3 per cent. of sulphur, and the anhydrous zinc sulphate contained 19.84 per cent. of sulphur. In all these tests the evolution of sulphurous acid was comparatively rapid when a decomposition-stage was reached, and then slowed down until it finally ceased altogether.

The preliminary tests having shown that with low temperatures the evolution of sulphurous acid gas was excessively slow, the subsequent tests were carried on at higher temperatures and with shorter intervals between temperature-raisings; further, the residue was weighed at the end of each heat, and thus the percentage-loss by volatilization was established. Five runs were made to study the general behavior of the salt unto complete decomposition, the rate of decomposition at certain temperatures, and the effect of the thickness of the charge. Table IV. gives the losses in weight obtained in raising the temperatures from 528° to 836° C. in the stated time intervals. The loss in weight is very small at first; it increases, however, with the temperature. Six hours' heating at 755° C. causes a loss in weight (8.62 per cent.) far greater than 72 hours' heating at or slightly above 610° C. in the preceding run (7.5 per cent.). After having been kept for 24 hours at 755° C., the loss in weight has fallen to 1.81 per cent. (the lowest decomposition temperature is 739° C., as will be shown later on), while the total loss amounts to only 22.23 per cent. As this progress was too slow, the temperatures were raised more quickly.

Table V. gives the results of the tests made at 836° C., the highest temperature reached in Table IV., varying the thickness of the charge by spreading the same weight of ore over a 3-inch and a 6-inch boat, which gave layers 0.25 in. and $\frac{1}{16}$ in. in thickness. The 3-in. boat was heated in the tube-furnace, the 6-in. boat, in the retort-furnace. The results are plotted in curves **A** and **B** of Fig. 2.

Table VI. gives the results of the tests made at 803° and 770° C., the salt having been spread in a 3-in. boat to a thickness of $\frac{1}{16}$ in. The results are plotted in curves **C** and **D** of Fig. 2.

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TABLE IV.—*Losses in Weight by Heating Zinc Sulphate.*

[Charge : 1.9562 g. of zinc sulphate in a 3-in. boat.]

Time of Each Heat.	Temperature.	Loss of SO_3 in Each Heat.	Total Loss of SO_3.
Hours.	Degrees Centigrade.	Per Cent.	Per Cent.
1	528	0.04	0.04
4	618	0.46	0.50
6	636	2.19	2.69
6	755	8.62	11.31
6	755	6.95	18.26
6	755	2.16	20.42
6	755	1.81	22.23
6	760	2.27	24.50
3	768	1.67	26.37
3	785	2.72	28.11
3	792	4.83	32.94
3	803	5.50	38.44
3	815	5.08	43.52
3	826	4.71	48.23
3	836	1.27	49.50

TABLE V.—*Losses in Weight of Zinc Sulphate Heated to* 836° *C.*

Charge : 2.255 g. $ZnSO_4$ in a 3-in. Boat.			Charge : 2.259 g. $ZnSO_4$ in a 6-in. Boat.		
Time.	Loss SO_3.	Loss S (Calculated).	Time.	Loss SO_3.	Loss S (Calculated).
Hours.	Per Cent.	Per Cent.	Hours.	Per Cent.	Per Cent.
1	13.60	5.44	1	19.78	7.91
2	18.57	7.43	2	26.37	10.55
3	20.68	8.27	3	32.70	13.08
4	23.14	9.26	4	38.98	15.59
5	25.79	10.28	5	44.58	17.83
6	27.64	11.07	6	47.74	19.10
7	30.04	12.02	7	48.64	19.46
8	32.09	12.84	8	49.11	19.64
9	34.06	13.62	9	49.50	19.80
10	35.90	14.36	10	49.61	19.84
11	37.93	15.17	11	49.79	19.91
12	39.72	15.88	12	49.96	19.98
13	41.45	16.58			
14	42.83	17.13			
15	44.16	17.66			
16	45.10	18.04			
17	46.00	14.40			
18	46.59	18.64			
19	46.98	18.79			
20	47.81	19.12			
21	48.59	19.44			
22	48.86	19.54			
23	49.33	19.73			
24	49.62	19.85			
26	49.77	19.90			

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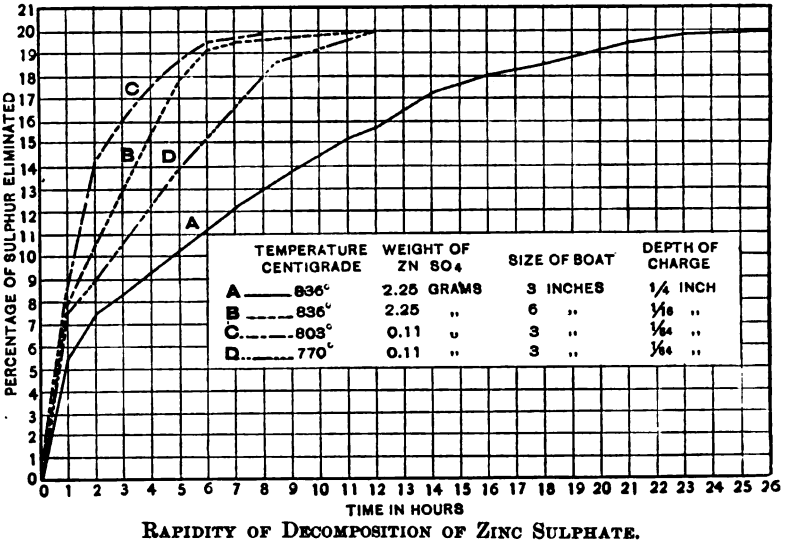
FIG. 2.


PERCENTAGE OF SULPHUR ELIMINATED
TIME IN HOURS
TEMPERATURE CENTIGRADE
WEIGHT OF ZN SO4
SIZE OF BOAT
DEPTH OF CHARGE
A ____ 836° 2.25 GRAMS 3 INCHES 1/4 INCH
B ____ 836° 2.25 " 6 " 1/16 "
C ____ 803° 0.11 " 3 " 1/64 "
D ____ 770° 0.11 " 3 " 1/64 "

RAPIDITY OF DECOMPOSITION OF ZINC SULPHATE.

TABLE VI.—*Losses in Weight of Zinc Sulphate Heated to 803° C. and 770° C.*

Temperature, 803° C.			Temperature, 770° C.		
Charge : 0.109 g. of $ZnSO_4$ in a 3-in. Boat.			Charge : 0.1179 g. of $ZnSO_4$ in a 3-in. Boat.		
Time.	Loss SO_3.	Loss S (Calculated).	Time.	Loss SO_3.	Loss S (Calculated).
Hours.	Per Cent.	Per Cent.	Hours.	Per Cent.	Per Cent.
1.....	20.18	8.07	1.....	18.82	7.53
2.....	35.59	14.24	2.....	22.30	8.92
3.....	40.00	16.00	3.....	26.29	10.51
4.....	43.95	17.58	4.....	30.45	12.18
5.....	46.61	18.64	5.....	34.61	13.84
6.....	48.63	19.45	8.5.....	46.40	18.56
7.....	49.09	19.64	11.5.....	49.37	19.57
8.....	49.55	19.82	12.....	49.96	19.98

Comparing curves **A** and **B** in Fig. 2, the importance of the thickness of charge upon the elimination of sulphur is brought out clearly. The charge and the temperature are the same in both cases. With a charge 0.25 in. thick (curve **A**), it takes 26 hours for complete desulphatization, while 12 hours (curve **B**) are sufficient to do the same work, if the charge has a depth of $\frac{1}{16}$ in. The loss in sulphur during the first hour's heating

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is rapid in both instances, but less so with the thicker charge (5.44 per cent.) than with the thinner one (7.91 per cent.); it then decreases gradually, again more rapidly with the thin than the thick charge. At the end of 6 hours the thick charge has lost 11.07 per cent. of sulphur, while with the thin charge 19.1 per cent. or nearly all the sulphur has been eliminated, yet it requires 6 hours more to drive off the remaining traces. When, after 12 hours, the thin charge has been freed from sulphur, the thick charge has lost only 15.88 per cent., and it requires 14 hours more heating to remove the remaining 4 per cent. of sulphur. Curves **C** and **D**, representing the heating of a small amount of salt at 803° and 770° C., are both steep; they show again the advantage of working with a thin charge and bring out clearly the effect of difference of temperature. With 803° C., the decomposition takes 8 hours, with 770° C., 12 hours. In the first hour the loss in sulphur is nearly the same, 8.07 per cent. (803° C.) and 7.53 per cent. (770° C.); the curves then diverge, the high-temperature curve rising quite rapidly as against the one of lower temperature. Comparing curve **B** (836° C., thickness of charge $\frac{1}{16}$ in.) and curve **D** (770° C., thickness of charge $\frac{1}{64}$ in.), it is seen that both charges are completely decomposed after 12 hours' heating; the elimination of sulphur at the higher temperature is more rapid in the first 6-hours' than at the lower temperature. It requires, however, 6 additional hours to remove the last 0.88 per cent. of sulphur at 836° C. with a $\frac{1}{16}$-in. layer, while only 2 hours are necessary to do the same work at 770° C. with a $\frac{1}{64}$-in. layer.

It ought to be added that in the runs represented by curves **A, B, C** and **D**, the heating was carried on 1 hour longer than shown in the tables and in the figure to have proof positive that no further loss took place.

(*c.*) *End of Decomposition.*—In the experiments recorded in Tables V. and VI. and represented graphically in Fig. 2, anhydrous zinc sulphate was completely decomposed at 770°, 803° and 836° C. It was of interest to find whether 770° C. was the lowest temperature of complete decomposition or not. For this purpose the experiments recorded in Table VII. were carried out. The charge of 0.1169 g. of dehydrated zinc sulphate in a 3-in. boat, was brought quickly to 763° C. and maintained for

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4 hours at this temperature. This was done to hasten the elimination of sulphur; weighings were made hourly to note the progress. The temperature was then lowered a few degrees at a time at half-hour intervals until no further loss in weight occurred. This took place at 730° C. The temperature was now raised to 735° C. and held there for 5.5 hours, half-hourly weights being taken.

TABLE VII.—*Losses in Weight of Zinc Sulphate Heated Rapidly to 763° C.*

[Charge: 0.1169 g. of zinc sulphate in a 3-in. boat.]

Time of Each Heating	Temperature.	Loss in Weight.	Loss of SO_3.
Hours.	Degrees Centigrade.	Grams.	Per Cent.
4.....	763	0.0310	26.52
0.5.....	752	0.0020	1.71
1.....	748	0.0032	2.74
0.5.....	739	0.0016	1.37
0.5.....	730	0.0000	0.00
5.....	735	0.0132	11.29
0.5.....	735	0.0000	0.00
1.....	777	0.0033	2.82
1.....	737	0.0013	1.11
0.5.....	737	0.0000	0.00
0.5.....	739	0.0014	1.20
0.5.....	739	0.0004	0.34
0.5.....	739	0.0000	0.00
0.5.....	749	0.0000	0.00
1.5.....	754	0.0000	0.00
0.5.....	842	0.0000	0.00

As there still remained in the residue 5.47 per cent. of sulphur trioxide, the temperature was brought to and maintained at 777° C. for 1 hour to drive off the bulk of the 5.47 per cent. Table VII. shows that 2.82 per cent. was thus removed. Lowering the heat to 737° C. and holding it there for 1 hour caused a loss of 1.11 per cent., but an additional half hour caused no further loss, although there still remained 1.54 per cent. of acid. Raising the temperature 2° to 739° C. caused in half an hour a loss of 1.2 per cent., and of 0.34 per cent. in the next half hour; a third half hour at 739° C. caused no further loss in weight, nor did heating to 749°, 754° and 842° C. The decomposition had been completed at 739° C.; the total loss in weight was 49.1 per cent.; the time 14 hours. The time required for decomposing the salt completely at 739° C. was not ascertained, but, judging by the runs at 803° and 770° C., it would have been of long duration.

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(*d.*) *Retardation in Heating-Curve.*—All the experiments so far carried out point to the conclusion that stable anhydrous basic zinc sulphates do not exist; the sulphur trioxide of the normal sulphate passes off gradually without any decided interruption, as the temperature is raised from beginning to complete decomposition. In order to verify this conclusion, the method employed by Bradford[11] in finding the temperatures of decomposition of ferrous, cupric and argentic sulphate was applied to zinc sulphate. It consists in heating the salt slowly in contact with a thermo-couple to and above decomposition and recording the rise of temperature. Upon dissociation of the substance there will be caused a break in the heating-curve; it will be a retardation if heat is absorbed, an acceleration if set free. Zinc oxide combining with sulphur trioxide forms an exothermic reaction,

$$ZnO + SO_3 = ZnSO_4 + 52.9 \text{ Kg. cal.};^{12}$$

hence, upon being decomposed, heat is absorbed, and the heating-curve will show a decided retardation, if all the sulphur trioxide is given off readily at a certain temperature, or if the normal anhydrous salt passes through clearly defined stops of basic salts while it is being converted into oxide.

The retort-furnace was used in the tests. It was placed on end and the retort partly filled with magnesia; a porcelain crucible charged with 5 g. of anhydrous zinc sulphate was embedded up to the rim in the magnesia. The junction of the thermo-couple was placed in the salt so as to nearly touch the bottom of the crucible. The current of 110 volts was turned on with the resistance-coils cut out. This gave at first a rise in temperature of 20° C. per minute, but it decreased, of course, as the temperature of the crucible rose. The galvanometer was read every 15 seconds; the readings were begun at little below 528° C. and continued to above 740° C. Four separate runs were made, three with the salt in the crucible, one with the empty crucible to serve as a blank test. In the plot, shown in Fig. 3, the galvanometer deflections and corresponding temperatures in degrees centigrade scale are plotted on the ordinate and the time in minutes on the abscissa. Curve A represents the blank

[11] *Trans.*, xxxiii., 50.

[12] Berthelot, *Thermo-Chimie*, Paris, 1897, vol. ii., pp. 91, 305, 309.

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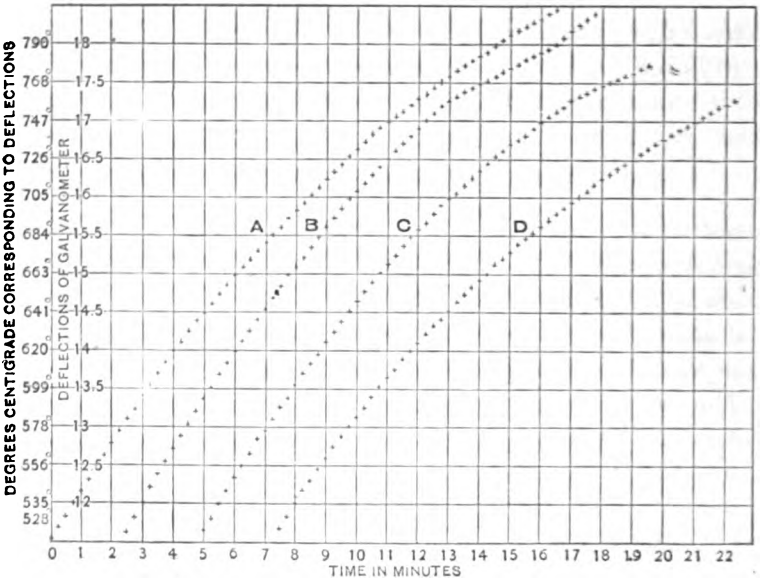
test, curves **B, C** and **D** the runs made with the charged crucible. Curve **A** shows slight retardations near 747° and 790° C., which are due to irregularities of current, caused probably by current being turned on or shut off in other parts of the Institute's laboratories. The apparent retardations in curve **B** at 760° C. and in curve **C** at 735° and 765° C. must be due to the same cause as that in curve **A**, as in curve **D** no retardation whatever is perceptible. These results therefore substantiate the conclusions formed from the preceding work.

FIG. 3.



HEATING-CURVE OF ZINC SULPHATE.

(*e.*) *Formation of Basic Salt.*—When anhydrous zinc sulphate, partly decomposed by heating, is treated with hot water, the undecomposed salt is quickly hydrated and dissolved, leaving behind basic salt, which dissolved with the greatest difficulty in water with continued boiling, small samples taking days of treatment. The following experiments were planned to see how much basic sulphate was formed as the decomposition of the normal sulphate and the elimination of sulphur progressed. The heating was carried on in the gas muffle-furnace. A charge of 80 g. of anhydrous normal zinc sulphate was placed in an oblong clay roasting dish, 4.25 by 10.25 in. and 0.75 in. deep, with walls

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0.25 in. thick. The temperatures were taken at intervals of a few minutes by placing the thermo-couple in contact with the salt. As the top of this was found to be 10° C. lower than the bottom, the couple was pushed down toward the bottom of the tray. The charge, 0.5 in. in thickness, was rabbled (with a few exceptions to be noted later) at intervals of a few minutes, and samples weighing from 10 to 14 g. were taken every hour. After a sample had been obtained, the salt was pushed together so as to have the same depth as the original charge. Three heatings were made: at 790° C., or 50° C. above the lowest temperature of complete decomposition of zinc sulphate; at 740° C., the temperature of decomposition; and at 690° C., 50° C. below the final temperature. It was necessary to make on each sample three analytical determinations: the total zinc, the zinc soluble in hot water, and the sulphur insoluble in hot water.

The total sulphur eliminated was calculated by deducting from the sulphur in charge, the insoluble sulphur and the sulphur in combination with the soluble zinc sulphate. Zinc was determined by titrating with potassium ferrocyanide and the residual sulphur by dissolving the basic sulphate in the smallest possible quantity of hydrochloric acid and then precipitating with barium chloride. The total zinc-content of each sample was ascertained in order that the analytical results might be reduced to a common basis, namely—to the basis of 40.52 per

TABLE VIII.—*Basic Zinc Sulphate Formed by Heating Zinc Sulphate.*

[Charge: 80 g. of $ZnSO_4$; temperature, 790° C.; time, 6 hours.]

	Actual Analyses.					
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc, total	43.44	48.53	48.97	49.49	52.05	54.49
Zinc as normal sulphate	36.28	26.40	25.73	24.62	22.02	19.82
Sulphur as basic sulphate	0.80	2.65	2.66	2.77	2.85	2.95
Analyses Corrected to Basis of 40.52 Per Cent. of Zinc.						
Zinc as normal sulphate	33.84	22.01	21.29	20.16	17.38	14.72
Sulphur as basic sulphate	0.747	2.21	2.20	2.27	2.25	2.14
Sulphur eliminated (calculated)	2.69	7.05	7.42	7.91	9.32	10.76

Average ratio: $\frac{\text{Sulphur eliminated}}{\text{Sulphur as basic sulphate}} = 3.82.$

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cent. of zinc contained in the normal anhydrous sulphate. The results of the actual and corrected analyses, and of the calculated corrected figures for eliminations of sulphur, are given in Tables VIII. (790° C.), IX. (740° C.) and X. (690° C.); the corrected figures are plotted in Fig. 4.

TABLE IX.—*Basic Zinc Sulphate Formed by Heating Zinc Sulphate.*

[Charge: 80 g. of $ZnSO_4$; temperature, 740° C.; time, 6 hours.]

	Actual Analyses.					
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc, total	40.82	42.27	43.27	44.83	45.74	46.55
Zinc as normal sulphate	38.90	36.88	35.69	33.29	31.09	29.20
Sulphur as basic sulphate	0.189	0.548	0.758	1.10	1.39	1.75
Analyses Corrected to Basis of 40.52 Per Cent. of Zinc.						
Zinc as normal sulphate	38.62	35.35	33.41	30.07	27.54	25.42
Sulphur as basic sulphate	0.188	0.525	0.709	0.997	1.23	1.53
Sulphur eliminated (calculated)	0.762	2.06	2.85	4.23	5.26	6.02

Average Ratio: $\frac{\text{Sulphur eliminated}}{\text{Sulphur as basic sulphate}} = 4.08.$

TABLE X.—*Basic Zinc Sulphate Formed by Heating Zinc Sulphate.*

[Charge: 80 g. of $ZnSO_4$; temperature, 690° C.; time, 6 hours.]

	Actual Analyses.					
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Zinc, total	40.53	40.60	40.63	41.03	41.11	41.37
Zinc as normal sulphate	39.69	39.28	38.84	38.47	37.91	37.40
Sulphur as basic sulphate	0.088	0.112	0.094	0.194	0.297	0.324
Analyses Corrected to Basis of 40.52 Per Cent. of Zinc.						
Zinc as normal sulphate	39.68	39.20	38.72	37.98	37.36	36.72
Sulphur as basic sulphate	0.088	0.112	0.093	0.191	0.293	0.317
Sulphur eliminated (calculated)	0.33	0.55	0.81	1.08	1.29	1.58

Average Ratio: $\frac{\text{Sulphur eliminated}}{\text{Sulphur as basic sulphate}} = 5.4.$

The values for total zinc show readily how far the decomposition of the normal sulphate was carried. Starting with 40.52

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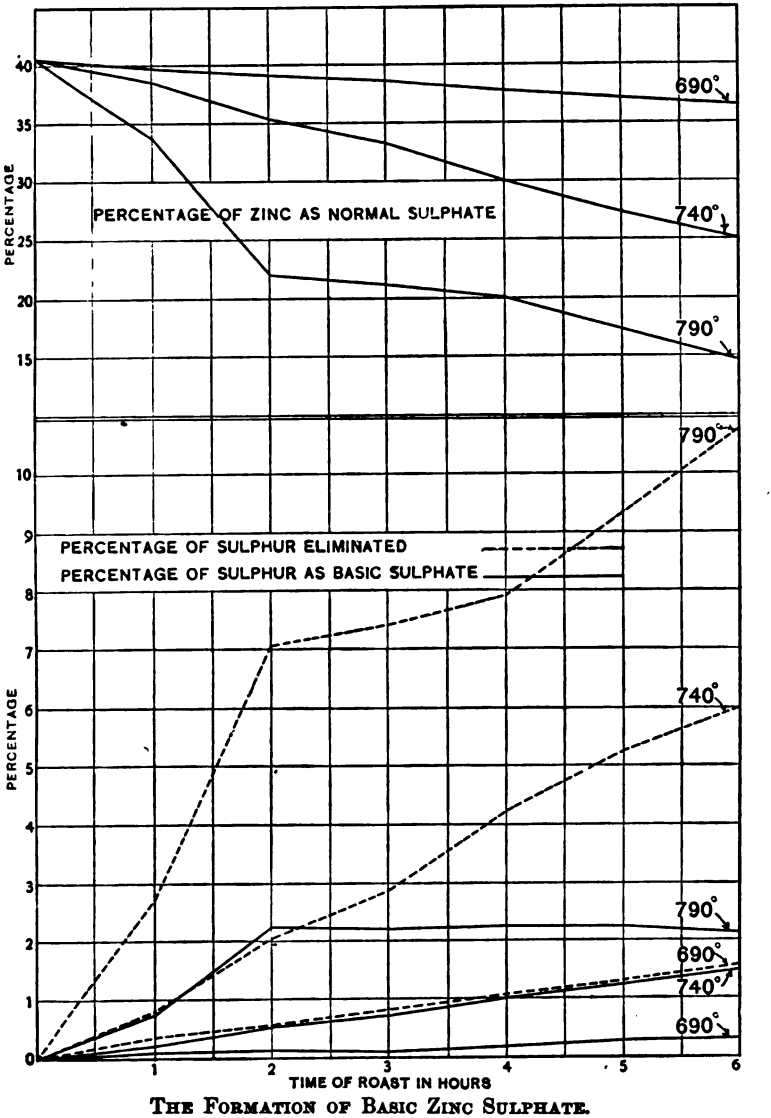
per cent. of zinc in the original charge, the percentage has risen at the terminations of the tests, with 790° C. to 54.59,

FIG. 4.



THE FORMATION OF BASIC ZINC SULPHATE.

with 740° C. to 46.55, and with 690° C. to only 41.37. The elimination of sulphur with 790° C. is rapid at first, slackens and then again increases; with 740° and 690° C. it is more

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uniform. The total proportion driven off is, of course, greater with the higher than with the lower temperature. A proportionality between the amount of sulphur eliminated and the degree of heat is indicated by the results at the ends of the roasts. The elimination of 10.76 per cent. at 790° C. fell to 6.02 per cent. at 740° C. and to 1.58 per cent. at 690° C.; or in a 6-hour heat a difference of 50° C. in temperature made a difference of 4.5 per cent. in the sulphur elimination.

In the tube-furnace, treating small amounts at a time, it had been noticed that at any given temperature the decomposition was at first rapid and then proceeded more slowly. In the muffle-furnace this was not the case, the difference being no doubt due to the stirring of the charge.

The percentage of sulphur present as basic sulphate increases with the high temperature of 790° C. for the first 3 hours, remains constant up to the fifth hour and then begins to decrease; with the lower temperatures of 740° and 690° C. it shows a steady increase. At the end of the heating the remaining percentage is higher with 790° than with 740° or 690° C. The average ratio of sulphur eliminated to sulphur as basic sulphate increases as the temperature decreases; it is 3.82 with 790°, 4.08 with 740° and 5.40 with 640° C. The percentage of normal zinc sulphate decreases as the temperature rises. With 790° C. there is a quick fall during the first 2 hours, during the next 2 hours it is slow, and accelerates again the last 2 hours. With 740° and 690° it is more uniform and, of course, greater with 740° than with 690° C.

PART II. DECOMPOSITION OF ZINC SULPHATE BY HEATING WITH CARBON (REDUCING-ROAST).

I. *Introduction.*

If metallic sulphates that are decomposed by heat alone are heated with a carbonaceous reducing agent they can be partly reduced, the sulphur trioxide oxidizing the carbon. This may be expressed by the equation

$$2\,MSO_4 + C = 2\,MO + 2\,SO_2 + CO_2.$$

If the metal has more than one oxide, the higher oxide may be brought to a lower state of oxidation.

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$$4\ MSO_4 + 3\ C = 2\ M_2O + 4\ SO_2 + 3\ CO_2.$$

If the metallic oxide is readily reducible, it may be partly brought to the metallic state and, combining with the sulphur, obtained by the reduction of sulphur dioxide, form a metallic sulphide.

$$2\ MSO_4 + C = 2\ MO + 2\ SO_2 + CO_2,$$
$$2\ MO + C = 2\ M + CO_2,$$
$$2\ SO_2 + 2\ C = 2\ S + 2\ CO_2,$$
$$2\ M + 2\ S = 2\ MS.$$

If the temperature is sufficiently high, the metallic sulphate is reduced to sulphide direct.

$$MSO_4 + 2\ C = MS + 2\ CO_2.$$

Reducing-roasts are not frequent now in connection with metallic sulphates; occasionally they are practiced in roasting blende carrying small amounts of galena, as the ready fusibility of galena and of some of its compounds makes it inadvisable to raise the charge to the temperature necessary for driving off all, or at least most, of the sulphur trioxide.

The only temperature measurements made in a reducing-roast are those by Scherr,[13] who found, upon addition of charcoal at 560° C. to partly roasted matte, that the 5 per cent. of soluble cupric sulphate was quickly reduced to 0.2 per cent. The first statement regarding zinc sulphate is that of Guy Lussac,[14] who investigated the behavior of the sulphates of zinc, iron, manganese, magnesium, nickel, lead, copper, silver and quicksilver in a reducing-roast. He found that if zinc sulphate, mixed with an excess of carbon over that required by theory, was brought in a closed vessel to a dark-red heat, there remained a residue of zinc oxide and carbon, while there passed off a gas consisting of 2 volumes of sulphur dioxide and 1 volume of carbon dioxide. If the mixture was brought quickly to a white heat, the residue was zinc sulphide and carbon, while the gas that passed off consisted of carbon monoxide, with a slight admixture of sulphur dioxide and carbon dioxide.

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[13] *School of Mines Quarterly*, 1900, xxi., 66.

[14] *Erdmann's Journal fuer Praktische Chemie*, 1837, xi., 65.

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II. *Experiments and Results.*

In the experiments with anhydrous zinc sulphate there were determined the temperatures at which the reaction began and at which it became vigorous, and the residue was analyzed for sulphur. Heating was carried on in the tube-furnace. As it was necessary to carry off the volatile products of the reaction, a current of gas had to pass through the tube. Air being excluded on account of the rapid combustion of the carbon, nitrogen was used. It was prepared by Lupton's method,[15] in which ammonia gas diluted with air is conducted over a mixture of cupric oxide and metallic copper heated to redness in a small ignition-tube. The metallic copper is oxidized by the air and again reduced by the hydrogen of the ammonia. The gas-mixture is obtained by forcing air through a concentrated solution of ammonia. It will be seen below, from the slow burning of the carbon in the tests, that the nitrogen thus obtained was not pure. As this did not interfere with the work, no especial pains were taken to prepare pure nitrogen. These runs were made with 6, 8 and 22 times as much carbon in the form of charcoal as was called for by the formula. The charge, about 0.4 g. of anhydrous zinc sulphate, was mixed and ground with the desired weight of charcoal and placed in a 3-in. porcelain-boat. The mode of operating was the same as in the previous tests made in the tube-furnace.

In the first test (6 times the theoretical quantity of carbon) the temperature of rapid reaction was first noted; it was 528° C. This is the temperature at which zinc sulphate begins to be decomposed when heated in air. The evolution of gas, however, was found to be much more energetic, showing that the decomposition by means of carbon at this temperature was more powerful than by means of heat alone. The temperature was then lowered; the evolution of sulphurous acid diminished correspondingly and ceased at 475° C. The residue was white, all the carbon had been oxidized. In run No. 2 (8 times the theoretical quantity of carbon) the lowest temperature at which any sulphur dioxide could be obtained was 409° C.; it ceased after 1 hour's heating. Upon raising the temperature to 425° C., sulphur dioxide was again liberated, but weakly, and lasted also

[15] *Chemical News*, 1876, xxxiii., 90.

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only for 1 hour. Heating at 528° C. caused the reaction to show much activity; it lasted 13 hours; the residue was again found to be white. In order to check the results of the two preceding tests and to investigate the behavior between 425° and 528° C., a third run (22 times the theoretical quantity of carbon) was made. The first sulphurous gas was again detected at 409° C.; at 463° C. carbon acted upon the sulphate, but not strongly; the effect weakened quickly and was hardly perceptible after 2.5 hours; at 500° C. the reaction was fairly rapid, but weakened after a few hours; at 528° C. there was the usual marked increase in the evolution of gas; the reaction continued for 27 hours, weakening toward the end; all the carbon had been consumed. Upon a further addition of carbon and heating to 528° C. no further evolution of sulphur dioxide could be detected.

The weights taken in the runs were those of the anhydrous zinc sulphate, of the charcoal and of the residue. As the percentage of ash of the charcoal had been determined, the weight of the decomposed sulphate could be easily found. The residue was analyzed for sulphide and sulphate sulphur by the method advocated by Blair.[16] The residue was dissolved in a 250-cc. decomposition-flask in a small amount of hydrochloric acid, and the hydrogen sulphide evolved conducted through an alkaline solution of lead nitrate held in a 140-cc. Erlenmeyer beaker. The hydrochloric acid solution held the sulphate sulphur and the absorption-flask the lead sulphide precipitated by the hydrogen sulphide. The leading results of the three tests are brought together in Table XI.

TABLE XI.—*Decomposition of Zinc Sulphate by Heating with Carbon.*

Test Number	1.	2.	3.
Multiple of theoretical carbon	6	8	22
Time of heating—hours	7	13	27
Temperature—degrees Centigrade	475	528	528
Loss in weight of $ZnSO_4$—per cent	6.27	30.55	42.61
Sulphide S in residue of $ZnSO_4$[a]—per cent	trace	0.61	1.41
Sulphate S in residue of $ZnSO_4$[a]—per cent	17.80	6.85	2.06

[a] $ZnSO_4$ contains 19.86 per cent. of S.

[16] *The Chemical Analysis of Iron and Steel*, Philadelphia, 1901, p. 60.

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III. *Summary.*

The reaction of carbon upon zinc sulphate begins at 409° C., but is weak and lasts only a short time; it grows stronger at 425°, 463,° 475°, 500° C., but weakens perceptibly after a few hours; only at 528° C. does carbon appear to become decidedly active, but even here the effect is at best very slow and imperfect, as it is opposed by the direct reduction of zinc sulphate to zinc sulphide. Under the most favorable conditions the 19.86 per cent. of sulphur of zinc sulphate are reduced to 3.47 per cent.; of this total 1.41 per cent. is sulphide-sulphur and 2.06 per cent. sulphate-sulphur.

PART III. FORMATION OF ZINC SULPHATE BY ROASTING.

I. *Introduction.*

Blende in its deposits is usually associated with other metallic sulphides, such as galena, pyrite, marcasite, chalcopyrite, etc. If the minerals are sufficiently coarse, galena can be readily separated from blende by crushing and washing; pyrite and chalcopyrite, however, only very imperfectly. Magnetic concentration has within a few years come to the assistance of the common methods of ore-dressing, and does satisfactory work. (In another paper this question is treated from a laboratory-experimental point of view.) When the metallic sulphides are very intimately mixed, ore-dressing methods fail and chemical methods come into play. The earliest of the latter class has been to give the ore a sulphatizing-roast,[17] aiming to oxidize iron and copper, and to sulphatize the lead and zinc and remove the zinc by leaching; but this has always been unsatisfactory. The only locality where it is carried on on a working-scale is at the Herzog-Julius and Frau Sophieen works in the Harz mountains.[18] Here the leaching with water has to be supplemented by treatment with acid to dissolve basic zinc salts and zinc oxide. While no details are given as to the percentage of extraction, it may be inferred to be unsatisfactory from the fact that the slag of the blast-furnace in which the roasted and leached ore is

[17] Ingalls, *Engineering and Mining Journal*, May 3, 1902.

[18] Bräuning, *Zeitschrift fuer Berg-, Huetten- und Salinen-Wesen in Preussen*, 1877, xxv., 144; Kerl, *Mineral Industry*, 1895, iv., 83; Schnabel, *Handbuch der Metallhuettenkunde*, 2d ed., Berlin, 1904, p. 312; Ingalls, *Metallurgy of Zinc and Cadmium*, New York, 1903, 677.

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smelted assays 19.12 per cent. of zinc. It becomes, therefore, of much value to study the conditions under which the maximum of sulphate is formed in roasting blende.

Plattner[19] is probably the earliest writer who investigated the behavior of blende when subjected to an oxidizing roast. He says that, if blende, practically free from iron and in a finely divided state, is roasted with stirring at a dark-red heat, the oxidation will proceed slowly, and the sulphur dioxide evolved will be converted in part into sulphur trioxide, this process being favored by a thick bed of ore. Some of the sulphur trioxide will combine with the zinc oxide formed to yield normal sulphate and tetrabasic sulphate, and some will have an oxidizing effect upon the blende still undecomposed. If the roast is carried on at an elevated temperature and with a strong draft, causing free access of air, less sulphur trioxide will be formed and hence less zinc sulphate. If the temperature reaches the point at which the tetrabasic sulphate is decomposed, the normal sulphate cannot well form, and the result is a product containing only zinc oxide. The formation of a tetrabasic zinc sulphate has already been discussed.

The presence of iron in blende as an iromorphous sulphide is known to have a retarding influence upon the roasting of blende. Minor[20] has published interesting remarks on this point, treating four samples of blende with from 2 to 15 per cent. of Fe. His results are given in part in Table XII.

TABLE XII.—*The Retarding Influence of Iron Upon the Roasting of Blende.*

Raw Blende.		Percentage of Sulphur in Roasting Blende.					
No. of Test.	Iron.	1 Hour.	2 Hours.	3 Hours.	4 Hours.	5 Hours.	6 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1.....	2.33	10.32	2.86	1.80	1.71	0.95	0.08
2.....	13.12	24.33	21.95	20.65	17.02	2.15	0.18
3.....	2.06	14.69	6.27	1.23	0.22	0.16	0.06
4.....	15.94	27.25	24.30	18.25	11.94	4.27	0.04

The roasts are seen to have been carried very far. The behavior of the samples low in iron, Nos. 1 and 3, is normal, the

[19] *The Metallurgischen Roestprocesse*, Freiberg, 1856, pp. 87, 142.

[20] *Chemiker Zeitung*, 1889, 1602; *Berg- und Huettenmaennische Zeitung*, 1889, 446.

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percentage of sulphur falling off quickly in the first two hours, and then more slowly. With the blende high in iron, samples Nos. 2 and 4, the percentage of sulphur decreases very, very slowly, and then drops suddenly in the fifth hour; at the end of the sixth hour the blende with 15.98 per cent. (or nearly 16) of Fe contains less sulphur than any of the other three samples. An explanation for this peculiar behavior is not given.

With regard to ferruginous blende Jensch[21] makes the surprising statement that when roasted sweet, the sulphur still present is almost wholly combined with iron as monosulphide. This is contrary to the general belief based upon practical experience, and the fact that iron sulphide is more readily oxidized than zinc sulphide; contrary to the researches of Plattner,[22] who found that, under normal conditions, the iron was oxidized to ferric oxide, and that, when the temperature was very high and the access of air limited, magnetic oxide was readily formed; and contrary to the researches of Prost,[23] who finds that part of the iron oxide formed combines with zinc oxide (see Appendix, Part IV.). The results of Jensch are, however, given in full in Table XIII. to show that it is possible to roast dead an iron-bearing blende and leave part of the iron in the form of iron sulphide. Unfortunately, the details of operating are not given.

The progressive desulphurization in large-scale work is discussed in detail by Ingalls,[24] to which the reader may be referred.

II. *Description of Blende.*

The experiments on the formation of zinc sulphate were made with blende from Joplin, Mo., and from Warren, N. H.; some work was done with a sample from New Mexico. The analyses made are given in Table XIV.

The Joplin blende was massive; it had the characteristic brown color and resinous luster. It was chosen on account of its freedom from gangue and iron. The mineral from Warren,

[21] *Zeitschrift fuer Angewandte Chemie*, 1894, 50; *Berg- und Huettenmœnnische Zeitung*, 1894, 299.

[22] *Op. cit.*, p. 143.

[23] *Bulletin de l'Association Belge des Chimistes*, x., 246; through Ingalls, *Metallurgy of Zinc and Cadmium*, New York, 1903, 32.

[24] *Metallargy of Zinc and Cadmium*, New York, 1903, pp. 45 *et seq.*

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TABLE XIII.—*Jensch's Results of Roasting Ferruginous Blende.*

		Upper Silesia.			Sweden.			Erzgebirge.	Styria.	
		a.	b.	c.	a.	b.	c.	a.	a.	b.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
I. Raw blende.	Zn.	32.64	22.58	32.07	31.66	35.92	32.45	44.60	44.95	42.80
	Fe.	5.95	7.12	4.66	10.53	8.64	10.75	4.29	0.72	0.97
	Pb.	3.12	3.20	2.88	6.88	11.10	5.38	2.50	1.12	0.85
	Mn.	.....	.....	.....	.....	1.30	0.10	.....	.....	.....
II. Roasted blende.	Zn.	43.60	30.04	39.81	35.22	40.72	34.90	51.70	56.10	49.66
	Fe.	8.11	10.00	5.73	11.78	9.17	11.62	5.12	0.84	1.16
	Pb.	3.95	4.19	3.31	7.24	12.04	5.68	21.86	1.20	0.97
	Mn.	.....	.....	.....	.....	1.46	.....	.....	.....	.....
III. Leached* residue from II.	Zn.	0.42	0.30	0.58	0.26	0.46	0.18	0.22	.....	0.18
	S.	1.64	1.83	2.02	1.30	2.40	1.06	1.29	1.36	0.68
	ZnS.	0.62	0.45	0.87	0.39	1.69	0.27	1.33	.....	0.27
	S available for iron.	1.44	1.68	1.73	1.17	2.17	0.98	1.18	1.36	0.59
	FeS.	3.96	4.62	4.74	3.22	5.97	2.69	3.24	0.99	1.62
IV. Sulphur calculated as in combination with	Zn.	0.20	0.15	0.29	0.13	0.23	0.08	0.11	.....	0.09
	Fe.	1.44	1.68	1.73	1.17	2.17	0.98	1.18	1.36	0.59
V. Proportion of	Zn : Fe.	1 : 7	1 : 11	1 : 6	1 : 9	1 : 10	1 : 12	1 : 11	1 : ∞	1 : 6.5

* Roasted blende (II.) was leached with water and ammonium acetate, to dissolve zinc and lead sulphate, and boiled with caustic soda, to dissolve zinc oxide and silicate, leaving as residue: zinc, iron and sulphur.

TABLE XIV.—*Analyses of Blende.*

	Joplin, Mo.	Warren, N. H.	New Mexico.
	Per Cent.	Per Cent.	Per Cent.
Zn	66.32	53.94	.....
Fe.....	0.45	8.80	13.40
S.....	32.54	31.37	.....
Pb.....	.....	3.82	.. ..
SiO_2.....	not determined	0.82	.....
Cd.....	not determined	trace	.....

N. H., was the best representative of blende high in iron that could be obtained in considerable quantity. The sample was picked over at the mine to obtain the ore as pure as possible. When crushed through an 8-mesh sieve in the laboratory, it was again hand-sorted. Nevertheless, some lead remained behind, as shown by the analysis. The ore was made up of crystalline fragments; it had a brownish-black color and a luster more adamantine than resinous. The blende from New Mexico, used in a few cases to see how it differed in its behavior from Warren blende, was a concentrate. The mineral was black and had an adamantine luster. It was carefully picked over to exclude all foreign matter.

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III. *Outline of Operations.*

The temperatures at which different specimens of blende decrepitate and ignite are of value in an investigation of the mineral; hence these were determined. The ore to be given a sulphatizing-roast was passed through 8- and 40-mesh sieves; a charge weighed from 200 to 350 g.; the roastings were carried on in the gas muffle-furnace; temperature and time were varied with the object of obtaining the maximum normal sulphate; the temperatures were measured as in the experiments for formation of basic salts (p. —); and samples for analysis were taken at intervals of from 0.5 to 1.5 hours. Four determinations were made from each sample: Total zinc, by titration with potassium ferrocyanide; total sulphur, by fusion with potassium nitrate and precipitation with barium chloride; normal zinc sulphate, by leaching with warm water (8 decantations of 20 cc. being sufficient) and titration of zinc as in the determination of total zinc; and sulphur in basic zinc sulphate, the method of Steinbeck[25] being employed (covering of sample with sodium bicarbonate, adding enough water for solution, allowing to stand for 24 hours with occasional stirring, filtering, acidifying and boiling filtrate, and precipitating with barium chloride). Beside the regular sulphatizing-roasts, special tests were made with Joplin blende by mixing it raw and dead-roasted with pyrite in order to increase the percentage of soluble sulphate. In this series all the samples taken were analyzed for normal zinc sulphate, and only those showing a maximum of soluble sulphate were analyzed for total zinc, total sulphur and sulphur as basic sulphate.

Attention has been called by Prost[26] to the probable existence of a zince ferrate, $ZnFe_2O_4$, formed in roasting ferruginous blende, which is insoluble in dilute sulphuric acid, and may be the cause of the unsatisfactory results obtained by sulphatizing-roasts. Several experiments were made to determine whether this compound existed, and, if this was found to be true, to find the conditions that favored its formation.

[25] "Chemisch-Analytische Untersuchungen über die Veraenderungen, welche der Mansfelder Kupferstein bei der Roestung erleidet." *Zeitschrift fuer Berg-, Huetten- und Salinen Wesen in Preussen*, 1863, xi., 123.

[26] *Op. cit.*

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IV. *Experiments in Detail.*

1. *Decrepitation.*—A small quantity of medium-size pieces was picked from each of the three lots of blende and crushed to pass an 8-mesh sieve. Each sample was then placed in a large boat and heated in the electric retort-furnace, with the junction of the thermo-couple in contact with the ore. The neck of the retort was kept open to permit watching the interior. The Warren blende did not decrepitate at all. The behavior of the blende from Joplin and from New Mexico was very similar, so they can be discussed together. Decrepitation began at 290° C. and lasted a few minutes. The temperature was then raised 10° at a time. With every rise decrepitation started again and did not cease when the ignition-point had been reached. When it had stopped, about 25 per cent. of the charge was very fine and the rest medium coarse, but many pieces had retained about their original size, i.e., they were caught on a 12-mesh sieve.

2. *Ignition-Point.*—Three or more tests were made in the electric tube-furnace with each sample to fix the ignition-point. The sample was placed in a boat, the mode of operating was the same as with the beginning of decomposition of zinc sulphate. In the preliminary tests, in which the furnace was heated rapidly to locate approximately the point, the ignition-temperature was found to be about 40° C. too high. In the regular tests the furnace was brought slowly to near the ignition-point and the temperature then raised at the rate of a degree to a degree and a half per minute. The tests on separate samples of decrepitated Joplin blende gave the ignition-points 480°, 479° and 482°, or an average of 480° C. Of the Warren blende, which did not decrepitate, 3 tests were made on coarse (through 8- and on 12-mesh sieve) and 3 on fine (through 80-mesh sieve) material. The tests on the former gave 557°, 557° and 556°, or an average 557° C. as the ignition-point; the latter, 518°, 513° and 515°, or as an average 515° C. The difference of 42° C. is, of course, due to the fine material being more easily heated through than the coarse. The blende from New Mexico gave 535°, 531° and 536°, or an average 534° C.

Table XV. shows that the ignition of ferruginous blende lies higher than when running low in iron, and that, as far as the three samples are concerned, the rise is nearly proportional to

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TABLE XV.—*Ignition-Point of Blende.*

Source of Blende.	Joplin.	Warren.	New Mexico.
Size of blende.....	coarse and fine	fine	coarse and fine
Percentage of iron.....	0.45	8.80	13.40
Ignition-point, degrees centigrade.....	480	515	534

the iron, it being 4.19° C. for an increase of 1 per cent. of iron. If the samples are to be roasted, they have to be brought to above their ignition-temperatures, as none of them appeared to have been decomposed. The light Joplin blende would have shown any outward change more readily than the two dark samples, but it looked just as bright and shiny as it did before it was heated. Something similar has been more recently noticed by Sharwood.[27]

3. *Sulphatizing-Roast of Joplin Blende.*—With the ignition-point of Joplin blende at 480° C., it was thought that a satisfactory start in sulphatizing could be made at a temperature about 200° higher. Three roasts were carried out in the gas muffle-furnace at 690°, 740° and 790° C., or 50° C. below, at, and 50° C. above, the lowest temperature of complete decompositlon of zinc sulphate. The results are given in Tables XVI. (690° C.), XVII. (740° C.) and XVIII. (790° C.).

With regard to Table XVI., the absence of data for basic salt during the first 6.5 hours is due to an error in manipulation, in that the samples were leached with sodium acetate after they had been freed from normal sulphate with warm water; from the remaining three samples the basic salt was extracted by treating with sodium bicarbonate. As the samples of ore were used up and the data would have no practical value, the experiment was not repeated.

The elimination of sulphur at this low temperature is regular, but very slow, the amount having fallen in 12.5 hours from 32.54 to 14.08 per cent. The percentage of normal sulphate increases gradually and reaches its maximum with 0.623 per cent., while the basic sulphate is still increasing, at the end of the test, after 12.5 hours.

[27] *Engineering and Mining Journal*, June 23, 1904.

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TABLE XVI.—*Results of Roasting Joplin Blende.*

[Analysis: Zn, 66.32; S, 32.54; Fe, 0.45 per cent. Charge: 350 g.; size, 8-mesh; temperature, 690° C.; time, 12.5 hours.]

	Actual Analyses.								
Sample taken after.....	Hrs. 1	Hrs. 2	Hrs. 3	Hrs. 4	Hrs. 5	Hrs. 6.5	Hrs. 8.5	Hrs. 10.5	Hrs. 12.5
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc	66.85	67.67	68.70	69.98	70.82	71.02	71.03	71.55	72.06
Total sulphur	30.35	28.47	25.68	23.71	21.52	19.42	19.03	17.48	15.30
Zinc as normal sulphate	0.155	0.199	0.206	0.290	0.335	0.409	0.463	0.672	0.638
Sulphur as basic sulphate	*a*	*a*	*a*	*a*	*a*	*a*	0.103 *b*	0.121 *b*	0.143 *b*
Analyses Corrected to the Basis of 66.32 Per Cent. of Zinc.									
Total sulphur	30.09	27.90	24.78	22.47	20.15	18.13	17.76	16.20	14.08
Zinc as normal sulphate	0.154	0.195	0.199	0.274	0.313	0.382	0.432	0.623	0.587
Sulphur as normal sulphate	0.077	0.098	0.100	0.137	0.156	0.191	0.216	0.312	0.294
Sulphur as basic sulphate	*a*	*a*	*a*	*a*	*a*	*a*	0.096 *b*	0.112 *b*	0.134 *b*
Sulphur as sulphide-sulphur	30.01	27.80	24.63	22.33	20.00	17.94	17.29	15.78	13.65

a Ore leached with sodium acetate which did not dissolve any basic sulphate; no sample left to repeat test with sodium bicarbonate.

b Ore leached with sodium bicarbonate, which decomposed basic sulphate.

c Calculated.

TABLE XVII.—*Results of Roasting Joplin Blende.*

[Analysis: Zn, 66.32; S, 32.54, Fe, 0.45 per cent. Charge: 350 g.; size, 8-mesh; temperature, 740° C.; time, 15 hours.]

	Actual Analyses.									
Sample taken after.....	Hrs. 1.5	Hrs. 3	Hrs. 4.5	Hrs. 6	Hrs. 7.5	Hrs. 9	Hrs. 10.5	Hrs. 12	Hrs. 13.5	Hrs. 15
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	70.54	73.08	74.43	74.98	74.87	74.92	74.44	74.49	74.61	76.17
Total sulphur.....	24.79	17.22	12.66	7.75	6.28	6.16	5.70	4.72	3.96	3.47
Zinc as normal sulphate.....	0.106	0.315	0.783	1.23	1.56	1.27	2.13	2.42	2.78	2.35
Sulphur as basic sulphate...	0.011	0.046	0.159	0.252	0.583	0.777	0.823	0.857	0.867	0.727
Analyses Corrected to the Basis of 66.32 Per Cent. of Zinc.										
Total sulphur.....	23.30	15.63	11.42	6.85	5.50	5.45	5.08	4.20	3.53	3.02
Zinc as normal sulphate.....	0.101	0.285	0.662	1.09	1.38	1.10	1.89	2.16	2.48	2.04
Sulphur (*a*) as normal sulphate.....	0.051	0.148	0.331	0.545	0.690	0.550	0.945	1.08	1.24	1.02
Sulphur as basic sulphate...	0.010	0.042	0.144	0.223	0.517	0.687	0.733	0.763	0.770	0.633
Sulphur (*a*) as sulphide-sulphur.....	23.24	15.44	10.94	6.08	4.29	4.21	3.40	2.36	1.52	1.37

a Calculated.

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Table XVII. shows that, with the higher temperature of 740° C., the decrease in total sulphur is more rapid than with 690° C., sinking to 4.2 per cent. after 12 hours, and to 3.02 per cent. after 15 hours; the fall is quicker at the beginning than at the end; the low percentage of sulphide-sulphur at the end of the roast shows that the blende has been well decomposed. On account of the larger amount of sulphur oxidized the percentage of normal sulphate found is also greater than before; with the exception of the time interval between the seventh and tenth hour the increase is regular and reaches a maximum after 13.5 hours. The basic salt forms less rapidly than the normal salt and attains its maximum at the same time as the normal salt.

TABLE XVIII.—*Results of Roasting Joplin Blende.*

[Analysis: Zn, 66.32; S, 32.54; Fe, 0.45 per cent. Charge: 350 g.; size, 8-mesh; temperature, 790° C.; time, 7 hours.]

	Actual Analyses.				
Sample taken after.....	1 Hour.	2.5 Hours.	4 Hours.	5.5 Hours.	7 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	66.90	75.82	76.07	76.10	77.20
Total sulphur.....	27.27	10.49	3.10	2.16	1.16
Zinc as normal sulphate.....	0.049	0.170	.....	1.38	0.93
Sulphur as basic sulphate.....	0.012	0.033	.....	0.760	0.480
Analyses Corrected to the Basis of 66.32 Per Cent. Zinc.					
Total sulphur.....	25.28	9.17	2.70	1.88	0.997
Zinc as normal sulphate.....	0.047	0.152	.....	1.20	0.799
Sulphur (a) as normal sulphate.....	0.024	0.076	.....	0.60	0.40
Sulphur as basic sulphate.....	0.011	0.029	.....	0.66	0.41
Sulphur (a) as sulphide-sulphur.....	25.24	9.06	.....	0.62	0.19

[a] Calculated.

Table XVIII. shows that, with a temperature of 790° C., the roast could be stopped after 7 hours, the total sulphur having fallen to 0.997 per cent. and the sulphide-sulphur to 0.19 per cent. As the roast was carried on above the temperature at which the normal sulphate is completely decomposed, the maximum soluble sulphate is reached early and the total is smaller than in the preceding test. That the basic sulphate should follow a similar path was to be expected.

A difficulty encountered in leaching some of the samples

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which were afterward discarded may be mentioned. They were treated for a long time with hot water, the waters remaining in contact with the ore for several hours before they were decanted; the dissolved zinc was determined in the decantations obtained in 24 hours. After leaching 5 days in this manner without removing all the soluble zinc, the ore was boiled for 36 hours with occasional decantations and still some zinc salt went into solution. The explanation is found in the fact that basic sulphates are slightly soluble in hot and boiling water.[28]

Comparing the three roasts as plotted in Fig. 5, it is seen that the elimination of total sulphur increases with the temperature, while the time required decreases. The curves for sulphide-sulphur (or undecomposed blende) keep close to the curves for total sulphur, remaining, of course, below them. The narrow space between a pair of lines shows how small is the quantity of sulphate formed. The largest amount of normal zinc sulphate readily soluble in water is obtained at 740° C.; leaving out the jog in the curve at the ninth hour, the progress is quite regular until the maximum is reached, but even at best the total of normal zinc sulphate is extremely small. Basic salts, practically insoluble in water, keep an even pace with the normal salt, more so with high (790° C.) and low (690° C.) temperatures than under the favorable conditions of 740° C.

4. *Sulphatizing-Roast of Warren Blende.*—The blende from Warren, N. H., contains 3.82 per cent. of lead. Part of the galena is so finely disseminated through the blende that it is impossible to remove it by picking, even when using a magnifying glass. The presence of 4.41 per cent. of galena complicates matters. It is known that in roasting galena slowly at a low temperature more sulphate is formed than in roasting quickly at a high temperature;[29] hence, as far as the galena is concerned, the latter is preferable. As 690° C., with easy-roasting blende, had given a very small percentage of soluble zinc sulphate, 790° and 900° C. were chosen as roasting temperatures. The amount of lead sulphate formed is liable to be increased[30] by

[28] Vogel, *Journal fuer Chemie und Physik*, edited by J. S. C. Schweigger, 1814, xi., 416.

[29] Hofman, *Metallurgy of Lead*, New York, 1899, p. 12.

[30] Plattner, *Die Metallurgischen Roestprocesse*, Freiberg, 1856, p. 45.

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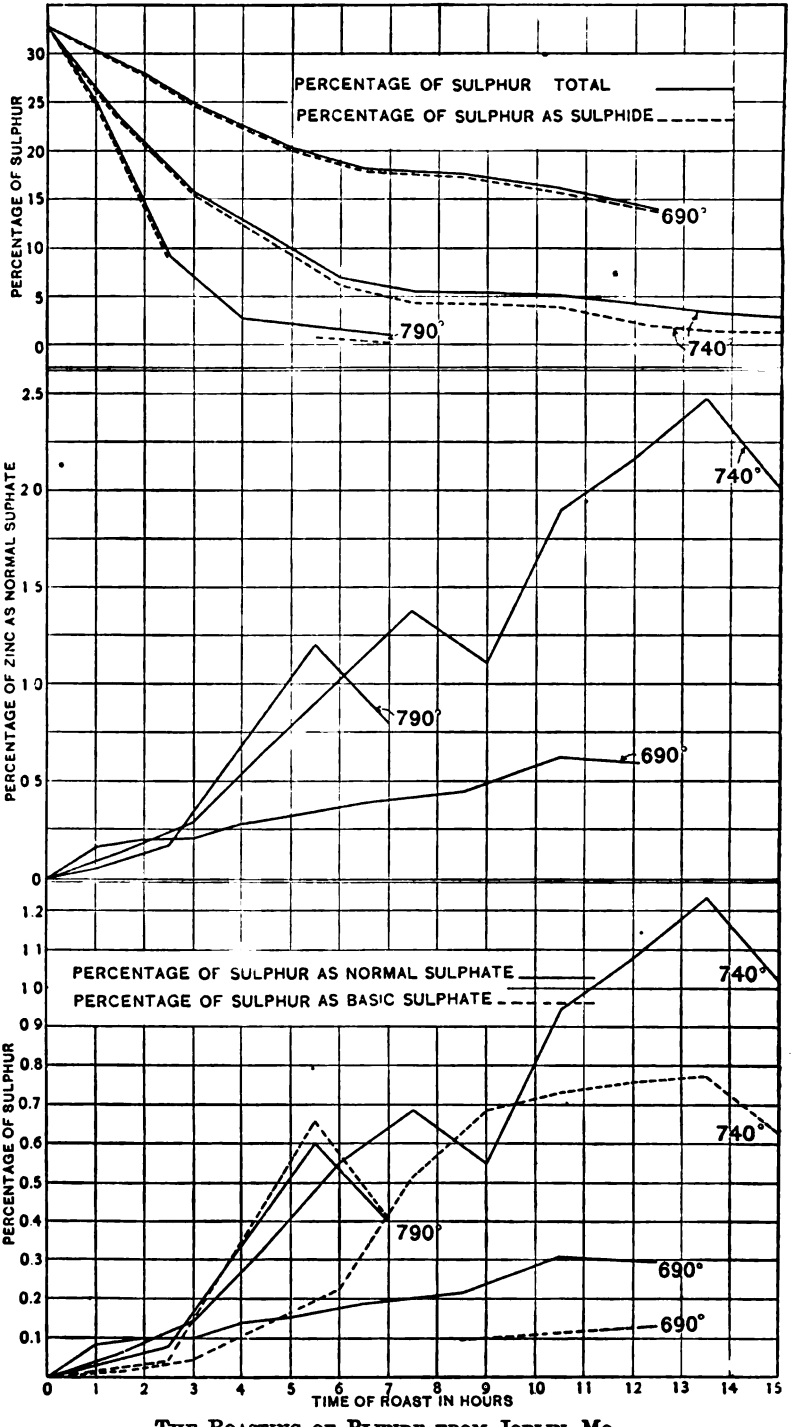
FIG. 5.



THE ROASTING OF BLENDE FROM JOPLIN, MO.

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the sulphur trioxide set free in roasting the blende. Assuming a quick roast to take place according to the formula

$$3\ PbS + 10\ O = 2\ PbO + PbSO_4 + 2\ SO_2,$$

it is fair to suppose that the proportion of $PbO : PbSO_4$ will be as 1:1 at 790° C. and slightly less at 900° C. The results obtained in the two roasts are given in Tables XIX. (790° C.) and XX. (900° C.), and graphically represented in Fig. 6.

A study of the data in Table XIX. shows that the elimination of total sulphur is regular, i.e., quick at the beginning and slow later on, but unsatisfactory, as after 15 hours it has been reduced to only 5.61 per cent. The percentage of normal zinc sulphate formed rises very quickly, reaches a maximum after 7.5 hours and then falls very gradually. The percentages of normal and basic sulphate-sulphur apparently keep an even pace, showing that under favorable conditions for normal sulphate basic sulphate forms in considerable amounts. This explains the necessity for acid-treatment in the Harz mountains after leaching with water.

TABLE XIX.—*Results of Roasting Warren Blende.*

[Analysis: Zn, 53.94; S, 31.37; Fe, 8.80; Pb, 3.82; SiO_2, 0.81 per cent.; Cd, trace. Charge: 300 g.; size, 8-mesh; temperature, 790° C.; time, 15 hours.]

	Actual Analyses.									
Sample taken after..	Hrs. 1.5	Hrs. 3	Hrs. 4.5	Hrs. 6	Hrs. 7.5	Hrs. 9	Hrs. 10.5	Hrs. 12	Hrs. 13.5	Hrs. 15
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	58.56	56.15	56.36	56.28	55.97	55.82	55.71	55.09	55.37	55.24
Total sulphur.....	19.56	12.58	10.79	10.07	9.75	8.66	7.93	6.65	6.42	5.96
Zinc as normal sulphate.....	0.597	3.76	4.41	5.06	5.08	4.74	4.68	4.37	4.21	3.95
Sulphur as basic sulphate...	0.400	1.41	1.66	1.75	1.76	1.55	1.49	1.51	1.49	1.49
Analyses Corrected to the Basis of 53.94 Per Cent. Zinc.										
Total sulphur.....	18.02	12.09	10.31	9.65	9.40	8.37	7.68	6.51	6.25	5.61
Zinc as normal sulphate.....	0.550	3.61	4.22	4.85	4.89	4.58	4.53	4.27	4.10	3.72
Sulphur (*a*) as normal sulphate.....	0.275	1.81	2.11	2.43	2.45	2.29	2.27	2.14	2.05	1.86
Sulphur as basic sulphate...	0.367	1.35	1.59	1.68	1.69	1.49	1.45	1.48	1.46	1.40
Sulphur (*a*) as sulphide-sulphur.....	17.34	8.93	6.61	5.54	5.26	4.59	3.96	2.89	2.74	2.35

[a] Calculated.

The results in Table XX. show that at 900° C. the percentage of total sulphur falls in 3 hours from 31.37 to 1.16 per cent.,

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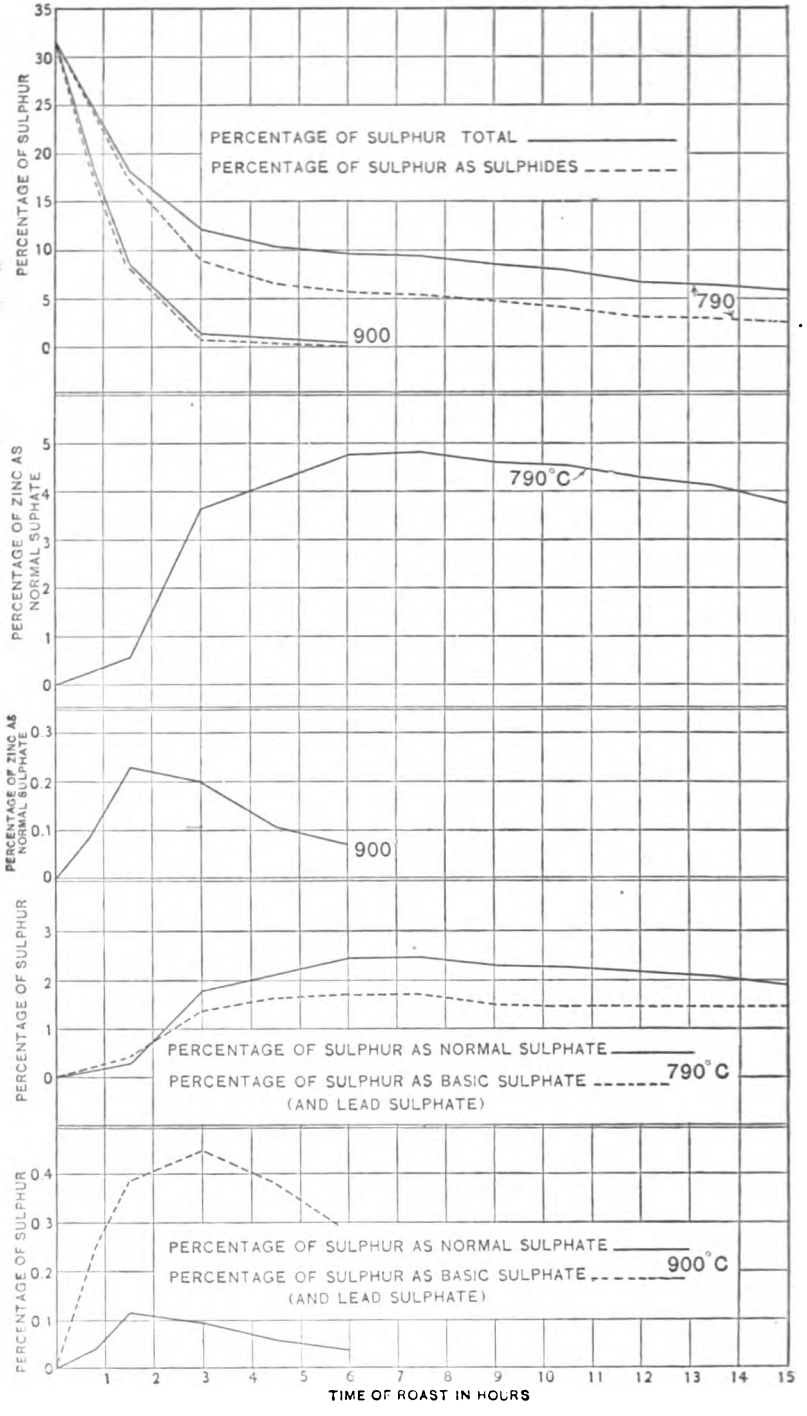
FIG. 6.



THE ROASTING OF BLENDE FROM WARREN, N. H.

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and is reduced in 6 hours to 0.48 per cent. (= 0.16 per cent. sulphide-sulphur). The amount of normal zinc sulphate formed at this high temperature must be low; the insignificant figure of 0.228 per cent. forms the maximum, and is reached after 1.5 hours. The evil effect of a high temperature is seen by the sulphur of the basic sulphate being always higher than that of the normal sulphate, increasing up to the third hour and then diminishing.

TABLE XX.—*Results of Roasting Warren Blende.*

[Analysis: Zn, 53.94; S, 31.37; Fe, 8.80; Pb, 3.82; SiO_2, 0.81 per cent.; Cd, trace. Charge: 300 g.; size, 8-mesh; temperature, 900° C.; time, 6 hours.]

	Actual Analyses.				
Sample taken after.....	0.75 Hrs.	1.5 Hours.	3 Hours.	4.5 Hours.	6 Hours.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total zinc.....	58.06	60.43	63.66	63.96	64.48
Total sulphur.....	19.74	9.62	1.37	0.98	0.58
Zinc as normal sulphate.....	0.085	0.255	0.232	0.125	0.082
Sulphur as basic sulphate.....	0.266	0.433	0.527	0.450	0.340
Analyses Corrected to the Basis of 53.94 Per Cent. Zinc.					
Total sulphur.....	18.35	8.59	1.16	0.83	0.48
Zinc as normal sulphate.....	0.079	0.228	0.196	0.105	0.068
Sulphur [a] as normal sulphate.....	0.040	0.114	0.098	0.053	0.034
Sulphur as basic sulphate.....	0.248	0.387	0.448	0.380	0.285
Sulphur [a] as sulphide-sulphur....	18.06	8.09	0.61	0.40	0.16

[a] Calculated.

A comparison of the two roasts in Fig. 6 shows for 790° C. clearly the imperfect elimination of total sulphur in 15 hours' roasting, coupled with a better formation of soluble zinc sulphate than is the case with 900° C. Lengthening the time of roasting to remove more sulphide-sulphur is unfortunately accompanied by a gradual but sure decomposition of the normal salt. This explains, perhaps, why it is that in the Harz mountains it has been found necessary to roast an ore repeatedly at a low temperature to obtain any commercial extraction whatever; a single prolonged roast would give unsatisfactory results. At 790° C. the basic sulphate curve is practically always beneath that of the normal sulphate; with 900° C. just the reverse is the case.

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Comparing the behavior of the Warren blende (Fig. 6) with that of the Joplin blende (Fig. 5), it is seen that in regard to the elimination of total sulphur, the behavior of Warren blende at 790° C. corresponds quite closely to that of Joplin blende at 740° C.; a resemblance is also to be found between Warren blende at 900° C. and Joplin blende at 790° C. In other words, the ferruginous blende gives up its sulphur less easily than the blende that runs low in iron. It may be that the geological age of the formation, as suggested by Jensch,[31] also has something to do with it, as Warren blende occurs in quartz-veins in Archaeic gneiss, while Joplin blende is found in Carboniferous limestone. The irony blende under suitable conditions furnishes more soluble zinc sulphate than blende low in iron, and the formation extends over a longer roasting period. This is to be expected, considering that iron sulphide is converted into normal ferrous sulphate which at 590° C. begins to be changed into ferric oxide, passing through stages of basic sulphate decomposed only at a higher temperature, all of which give off sulphur trioxide, which has an oxidizing effect upon zinc sulphide and a sulphatizing effect upon zinc oxide. The sulphatizing effect of basic iron sulphate is brought out also by the quick rate of decomposition of the soluble zinc sulphate with the Joplin blende when the maximum has been reached in comparison with the blende from Warren. In agreement with the larger amount of normal zinc sulphate formed with blende rich in iron stands the fact that it forms more basic zinc sulphate than blende low in iron.

5. *Sulphatizing-Roast with Addition of Pyrite.*—Starting with the fact proved by the preceding tests that a ferruginous blende gave more soluble zinc sulphate than blende low in iron, it was thought that an addition of pyrite to Joplin blende might greatly improve matters and furnish a satisfactory percentage of normal sulphate. Thus, a concentrate of galena, blende and pyrite, when subjected to a sulphatizing-roast, might furnish after leaching with water a desirable smelting ore of lead. Two tests were carried out, one with raw blende, the other with dead-roasted blende, to study the sulphatizing effect of pyrite under two opposite conditions. It was hoped that by furnishing to the sulphur trioxide a large quantity of zinc oxide on which it

[31] *Loc. cit.*

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could act, a considerable quantity of soluble-sulphate would be formed, as sulphur trioxide acts more strongly upon zinc oxide than upon zinc sulphide. The pyrite used came from the Davis mine in western Massachusetts. It was very pure, the analysis giving 52.95 per cent. of sulphur or only 0.44 per cent. less than that required by the formula. In considering the quantity of pyrite to be added to the blende, the plan was to have twice the amount required by theory, i.e., 2 parts of sulphur to 1 part of zinc, or 120 g. pyrite for 100 g. blende and for 83.8 g. zinc oxide. It was expected that one atom of sulphur of the pyrite starting to burn below the ignition-point of the blende would have practically no sulphatizing effect, but that the remaining monosulphide would accomplish the desired result. The temperatures chosen were 790° and 650° C., i.e., above and below 740° C., the temperature at which Joplin blende had given the maximum of soluble salt. Heating at 790° C., when zinc sulphate is rapidly decomposed, would show how much the sulphatizing effect of pyrite could counteract this tendency; the temperature of 650° C., high enough to decompose ferrous sulphate rapidly but zinc sulphate only very slowly, whether with very incomplete desulphurization of blende the oxidized part would be sulphatized by the pyrite. The results are recorded in Tables XXI., XXII. and XXIII.

The data in Table XXI. were obtained by roasting a mixture of blende and pyrite ground to 8-mesh size, the temperature being raised very slowly in order to avoid mechanical loss by decrepitation and to permit control of temperature when the pyrite began to burn. This occurred at about 500° C. The temperature rose in half an hour to 790° C., when the gas was shut off; it remained constant for several minutes and then began to fall, necessitating the turning on of the gas. With the ignition of the pyrite, dense white fumes arose and continued for half an hour, when the pyrite ceased to glow. The first sample was taken half an hour after the ore had reached 790° C.; the others as shown in the headings. The results in Table XXI. show a steady increase of soluble zinc sulphate, but, after 6.5 hours, the total had reached only 2.15 per cent. corrected weight; it could not have risen much higher, as the total sulphur had been reduced to 3.66 per cent.; hence the roast was not carried any farther. The total amount of sul-

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phate, however, is twice as large as when the raw blende was roasted alone at 790° C., and that of basic sulphate very much lower; both facts show that the addition of pyrite has a beneficent effect.

TABLE XXI.—*Results of Roasting Joplin Blende and Pyrite.*

[Charge: 100 g. Joplin blende and 120 g. Davis pyrite; temperature, 790° C.; time, 6.5 hours. Calculated analysis: Zn, 30.14; S, 43.68; Fe, 25.43 per cent.]

Size of ore, 8-mesh.

Sample taken after	0.5 Hours.	2 Hours.	3.5 Hours.	5 Hours.	6.5 Hours.
Zinc as normal sulphate	Per Cent. 0.42	Per Cent. 1.00	Per Cent. 2.19	Per Cent. 2.33	Per Cent. 2.70

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	37.83	4.59	2.70	1.35	0.831	2.41
Corrected to 30.14 per cent. of zinc	30.14	3.66	2.15	1.08	0.663	1.92

Size of Ore, 40-Mesh.

Sample taken after	0.5 Hours.	2 Hours.	3.5 Hours.	5 Hours.	6.5 Hours.
Zinc as normal sulphate	Per Cent. 1.32	Per Cent. 3.27	Per Cent. 3.18	Per Cent. 2.75	Per Cent. 1.79

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphate-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	39.92	2.85	3.27	1.64	0.868	0.34
Corrected to 30.14 per cent. of zinc	30.14	2.15	2.47	1.24	0.657	0.25

The second set of results in Table XXI. was obtained by roasting a mixture of blende and pyrite ground to pass a 40-mesh screen, the other conditions being exactly the same as in

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the first set of experiments. This fine grinding was made in order to have a finer material and a more intimate contact between particles than was possible with the previous charge of 8-mesh grains. The effect is striking in that the maximum normal sulphate, only slightly higher than with the coarser material (2.47 per cent. *vs.* 2.15 per cent.), is reached after 2 hours, and that the total sulphur at the same time reduced to a lower figure (3.66 per cent. *vs.* 2.15 per cent.). The ratios of sulphur as basic sulphate to sulphur as normal sulphate of the 8-mesh and 40-mesh samples at maximum normal sulphate do not differ much, viz., 0.663 : 1.08 = 0.61 for the coarse, and 0.657 : 1.24 = 0.53 for the fine; they are, however, about one-half of that obtained when the 8-mesh blende is roasted by itself, and show that in the latter case much more basic sulphate is formed than in the former.

The data in Table XXII. were obtained from a charge of 100 g. Joplin blende roasted dead in 2 hours at 950° C., the sulphur being reduced to 0.085 per cent. The roasted ore was cooled, the theoretical quantity (83.8 g.) weighed out and mixed with pyrite of 8-mesh size. The percentage of normal sulphate increased with the time of roasting. It reached, however, only 2.50 per cent. after 3 hours, when further heating was stopped. It could not have gone much higher by prolonging the heating, as the total sulphur had already been reduced to 2.13 per cent.

The conditions in this roast, given in the lower part of Table XXII., were the same as those in the upper part, excepting that the dead-roasted blende and pyrite had been crushed together to pass a 40-mesh sieve. The effect of fineness of ore and consequent intimate contact of particles is as apparent here as it was with raw blende and pyrite in Table XXI. The maximum of soluble zinc sulphate, 5.18 per cent., is reached after 2 hours with the total sulphur reduced to 3.57 per cent. and the sulphide-sulphur to 0.07 per cent. It is a great deal higher than with the 8-mesh mixture of Table XXII.; with that, some iron sulphide must have remained undecomposed, as the corrected analysis shows 0.57 per cent. of sulphur. The ratios of sulphur as basic sulphate to sulphur as normal sulphate (0.25 for 8-mesh, and 0.35 for 40-mesh, mixture) are lower than those of 790° C., or a smaller amount of basic sulphate is formed at

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TABLE XXII.—*Results of Roasting Dead-Roasted Blende and Pyrite.*

[Charge: 83.8 g. dead-roasted Joplin blende and 120 g. Davis pyrite; temperature, 650° C.; time, 3 hours. Calculated analysis: Zn, 32.54; S, 31.18; Fe, 27.43 per cent.]

Size of ore, 8-mesh.

Samples taken after	1 Hour.	2 Hours.	3 Hours.
Zinc as normal sulphate	Per Cent. 1.50	Per Cent. 2.48	Per Cent. 3.12

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	40.70	2.67	3.12	1.56	0.387	0.72
Corrected to 32.54 per cent. of zinc	32.54	2.13	2.50	1.25	0.308	0.57

Size of ore, 40-Mesh.

Sample taken after	1 Hour.	2 Hours.	3 Hours.
Zinc as normal sulphate	Per Cent. 5.82	Per Cent. 6.04	Per Cent. 5.50

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual	37.97	4.16	6.04	3.02	1.06	0.08
Corrected to 32.54 per cent. of zinc	32.54	3.57	5.18	2.59	0.906	0.07

a lower temperature. The normal, as well as the basic zinc sulphate found in these tests, contains a trace of iron sulphate.

In order to obtain another means of comparison for raw blende and roasted blende mixed with pyrite, a test was made with a 40-mesh mixture of raw blende and pyrite at 650° C. The total sulphur would not, of course, be reduced to a low

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figure; that which was set free might, nevertheless, have a strongly sulphatizing effect. The maximum soluble-sulphate was obtained after 8.25 hours, 4.13 per cent. by actual and 3.49 per cent. by corrected analysis. The results of this experiment are given in Table XXIII. This is the second highest figure obtained in the sulphatizing tests with blende- and pyrite-mixtures, although the total sulphur has been reduced only to 13.8 per cent. and the sulphide-sulphur to 12 per cent. The sulphide-sulphur will have been, in all probability, in combination with zinc.

TABLE XXIII.—*Results of Roasting Blende and Pyrite.*

[Charge: 100 g. Joplin blende and 120 g. Davis pyrite; temperature, 650° C.; time, 12.75 hours. Calculated analysis: Zn, 30.14; S, 43.63; Fe, 25.43 per cent.]

Size of ore, 40-mesh.

Samples taken after.....	Hrs. .75	Hrs. 2.25	Hrs. 3.75	Hrs. 5.25	Hrs. 6.75	Hrs. 8.25	Hrs. 9.75	Hrs. 11.25	Hrs. 12.75
Zinc as normal sulphate.....	Per Cent. 0.68	Per Cent. 1.93	Per Cent. 3.04	Per Cent. 3.30	Per Cent. 3.65	Per Cent. 4.13	Per Cent. 4.11	Per Cent. 4.10	Per Cent. 4.06

Analysis of Ore at Maximum Normal Sulphate.

	Zinc.	Sulphur.	Zinc as Normal Sulphate.	Sulphur as Normal Sulphate.	Sulphur as Basic Sulphate.	Sulphur as Sulphide-Sulphur.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Actual.....	35.68	16.34	4.13	2.07	0.05	14.82
Corrected to 30.14 per cent. Zn.....	30.14	13.80	3.49	1.75	0.04	12.00

The ratio of sulphur as basic sulphate to sulphur as normal sulphate, is 0.026; that of the two preceding tests, 0.25 and 0.35. The enormous decrease is probably due to the complete decomposition of basic iron sulphates in the 8.25 hours, while the 3 hours of the other two roasts were not sufficient for this purpose; in none of the samples taken after 3.75 hours' roasting could any dissolved iron be detected. Between 8.25 and 12.75 hours' roasting very little soluble zinc sulphate is decomposed, the difference, 4.13 — 4.06, being only 0.07 per cent.

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It ought to be stated that in all the roasts carried out at 650° C., the charge showed a tendency to rise above the given temperature as long as pyrite was burning freely. Whenever the pyrometer indicated a rise to the danger-point, the tray was removed from the muffle and allowed to cool to the desired degree of heat. After pyrite had ceased to burn, the temperature showed no tendency to rise above 650° C.

6. *Summary of Sulphatizing-Roasts.*—In order to bring together the leading features of the results obtained in the sulphatizing-roasts, the main points have been assembled in Table XXIV. The conclusions arrived at are:

1. The amount of normal zinc sulphate soluble in hot water obtained in the tests is very small. Based upon 100 parts of zinc in the charge, it reaches 9.07 per cent. with raw blende and 15.92 per cent. with dead-roasted mixed with a large excess of pure pyrite. Both extractions are too small to be of much practical value.

The fundamental difficulty in obtaining a satisfactory percentage of extraction lies in two facts: normal zinc sulphate is rapidly decomposed at the temperature for roasting blende; at the beginning of a roast, sulphur dioxide strongly predominates over sulphur trioxide. In the process carried out in the Harz mountains, the ore is roasted very slowly in heaps which have to be turned three times; the first heap of 500 tons ore burns from 6 to 7 months, the second fire lasts from 6 to 8 weeks, and the third from 4 to 6 weeks. Normal and basic sulphates are formed mainly in the fines, i.e., in the relatively cool cover, and these alone are leached with water and sulphuric acid. The probable explanation for the formation of these sulphates is that zinc sulphate formed toward the hotter center of a heap is decomposed, and the sulphur trioxide, dioxide and oxygen rising attack the fines in the cool cover and form normal and basic salts, or normal salts alone which are decomposed in part when the heat creeps up toward the end of the roast.

2. The ratio of sulphur as basic sulphate to sulphur as normal sulphate increases with the temperature.

3. Ferruginous (Warren) blende is more difficult to roast than blende running low in iron (Joplin), but the former gives under similar conditions of treatment a higher yield in normal sulphate owing to the presence of isomorphous sulphide, the

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TABLE XXIV.—*Summary of Sulphatizing-Roasts of Blende With and Without the Addition of Pyrite.*

Kind of ore.	Joplin Blende.			Warren Blende.		a. Raw Joplin Blende and Davis Pyrite.		b. Roasted Joplin Blende and Davis Pyrite		a. Raw Joplin Blende and Davis Pyrite.
Analysis of ore.	Zn, 66.32%. S, 32.54%. Fe, 0.45%.			Zn, 53.94%. S, 31.37%. Fe, 8.80%. Pb, 3.82%.		Zn, 30.14%. S, 43.68%. Fe, 25.43%.		Zn, 32.54%. S, 31.18%. Fe, 27.43%.		Zn, 30.14%. S, 43.68%. Fe, 25.43%.
Weight of roasting charge, grams.	350	350	350	300	300	220	220	203	203	220
Size of ore, mesh to linear inch.	8	8	8	8	8	8	40	8	40	40
Temperature of roast, °C.....	690	740	790	790	900	790	790	650	650	650
Time of roast, hours.....	12.5	15	7	15	6	6.75	6.75	3	3	12.75
Maximum normal zinc sulphate, after hours.	10.5	13.5	5.5	7.5	2.25	6.75c	2.25	3	2	7.25
Weight of leaching sample, grams.	10 and 3	10	10	10	10	3	3	3	3	3
Maximum normal zinc sulphate, per cent.	0.623	2.48	1.20	4.89	0.228	2.47	2.47	2.50	5.18	3.49
Maximum basic zinc sulphate, after hours.	d.	13.5	5.5	7.5	3.75	d.	d.	d.	d.	d.
At maximum normal sulphate. — Sulphur as normal sulphate, per cent.	0.312	1.24	0.60	2.45	0.114	1.08	1.24	1.25	2.59	1.75
At maximum normal sulphate. — Sulphur as sulphide-sulphur, per cent.	15.74	1.52	0.62	5.26	3.09	1.92	0.25	0.57	0.07	12.00
At maximum basic sulphate. — Sulphur as basic sulphate, per cent.	d.	0.77	0.66	1.69d	0.45d	d.	d.	d.	d.	d.
At maximum basic sulphate. — Sulphur as sulphide-sulphur, per cent.	d.	1.52	0.62	5.26	0.61	d.	d.	d.	d.	d.
At end of roast. — Sulphur as normal sulphate, per cent.	2.94	1.02	0.40	1.86	0.084	not determined.				
At end of roast. — Sulphur as basic sulphate, per cent.	0.134	0.633	0.41	1.40	2.85					
At end of roast. — Sulphur as sulphide-sulphur, per cent.	13.65	1.37	0.19	2.35	0.16					
At end of roast. — Sulphur, total, per cent.	14.08	3.02	0.997	5.61	0.48					
Zinc extracted, at maximum normal sulphate, per cent. on zinc in ore.	0.94	3.74	1.81	9.07	4.23	7.99	8.19	7.68	15.92	11.58

a. Raw Joplin Blende, 100 g.; Davis Pyrite, 120 g.
b. Roasted Joplin Blende, 83.8 g.; Davis Pyrite, 120 g.
c. Roast not carried to maximum soluble-sulphate.
d. Maximum not reached.

basic salts of which have a strongly sulphatizing effect when they are decomposed by heat.

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PART IV. APPENDIX: FORMATION OF ZINC FERRATE.

It has been proved that, in dead-roasting ferruginous blende under normal conditions, both the iron and zinc sulphides are converted into oxides. If these oxides were merely a mechanical mixture, it ought to be easy to remove all the zinc oxide by a solvent which did not attack the iron oxide, but in treating such ores it has been found that a considerable proportion of the zinc remained in the residue. Prost[32] thinks that zinc oxide and ferric oxide from zinc ferrate, $ZnFe_2O_4$, which is not readily soluble, and Ingalls also inclines to this belief.

While this question is not directly connected with the subject in hand, it seemed of interest to make a few synthetical tests in order to find out whether such a compound was formed or not. These tests consisted in heating weighed quantities of zinc oxide and ferrous sulphate, and of zinc oxide and ferric oxide, in a porcelain crucible over a Bunsen burner, and leaching the heated mixture with ammonium sesquicarbonate. A solution of this salt was chosen instead of the pure ammonia liquor, for the reason that tests made by me years ago[33] had shown that the sesquicarbonate was an excellent solvent for ignited zinc oxide, while pure ammonia was not.

The chemically pure zinc oxide used for the tests was found to have taken up carbonic acid by long standing; an analysis gave 68.48 per cent. of zinc instead of the theoretical 80.26 per cent. Crystals of green vitriol furnished the ferrous sulphate; the ferric oxide was prepared by roasting ferrous sulphate. In order to form $ZnFe_2O_4$, the proportions of Zn : Fe required are 54 : 93. The results of the series of tests made are given in Table XXV.

In making up the mixtures, the quantity of zinc was kept constant and that of the iron varied. The time of heating ranged from 1 to 2 hours; 1 hour, however, was sufficient to produce the desired result, as is seen from tests Nos. 4 and 5. Experiments in leaching proved that a time of longer duration than 2 hours gave no additional extraction, hence 2 hours was chosen as standard.

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[32] *Bulletin de l'Association Belge des Chimistes*, x., 246–263, through Ingalls, *Metallurgy of Zinc and Cadmium*, New York, 1903, p. 32.

[33] *Zeitschrift fuer Berg-, Huetten- und Salinen- Wesen in Preussen*, 1880, xxviii., 279 and 280.

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TABLE XXV.—*Results of Heating Zinc Oxide with Ferric Sulphate and with Ferric Oxide.*

Experiment, No.	1.	2.	3.	4.	5.	6.
Proportion by weight, zinc : iron	54 : 3	54 : 9	54 : 27	54 : 93	54 : 93	54 : 9
Time of heating, hours	1	1	2	1	2	1
Time of leaching, hours	2	2	2	2	2	2
Total zinc, leached, per cent	98.19	92.96	77.85	39.42	39.65	99.91
(a) Total zinc in residue, per cent	1.81	7.04	22.15	60.58	60.35	0.09
(b) Total zinc, per cent. of, required to form $ZnFe_2O_4$	3.22	9.67	29.00	100.00	100.00	9.67
Ratio, a : b	0.56	0.73	0.76	0.60	0.60	.....

In experiment No. 1 the proportion weight of zinc and iron is 54 to 3. Of the total zinc present, 98.19 per cent. was dissolved out, leaving 1.81 per cent. in the residue. If the compound $ZnFe_2O_4$ had been formed, 3.22 per cent. of zinc, instead of 1.81 per cent., ought to have remained insoluble. With the increase of iron in the mixture the quantity of zinc held back in the residue also grows. In experiments Nos. 4 and 5, all the zinc ought to have remained in the residue, but 39.42 and 39.65 per cent. was leached out. In experiment No. 6, in which zinc oxide was mixed with ferric oxide, practically all the zinc oxide was removed by ammonium sesquicarbonate, or no compound whatever was formed.

The tests, as far as they have been carried, tend to show that upon heating zinc oxide with ferrous sulphate various compounds of zinc and iron are formed, and not a single definite compound of the formula $ZnFe_2O_4$. The quantity of zinc remaining insoluble increases up to a certain point and then again decreases. Synthetical experiments with zinc oxide and ferrous sulphate cannot truly represent the process that takes place in roasting ferruginous blende; they can only point toward the direction of the reactions. The whole question of zinc ferrate must therefore be solved in a laboratory that is connected with central works treating blendic ores with different percentages of iron.

In dissolving the different samples of blende obtained in the sulphatizing-roasts, it was found that roasted Joplin blende dis-

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solved readily in a mixture of potassium chlorate and nitric acid, but roasted Warren blende did not, but left behind a black residue, which was only slightly attacked by warm nitrohydrochloric acid, but dissolved readily when brought to a boiling-heat.

NOTE BY THE SECRETARY.—Comments or criticisms upon all papers, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

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The Coal-Fields of Missouri.

BY B. F. BUSH, ST. LOUIS, MO.

(Lake Superior Meeting, September, 1904.)*

Area and Distribution.

The coal-fields of Missouri, situated in the northern and western portion of the State, are distributed, in whole or in part, over 57 counties, embracing an area estimated by Mr. Broadhead[1] to be practically 23,000 sq. miles, including small detached beds and outliers in St. Louis, St. Charles, Lincoln and Warren counties in the eastern part of the State. This area includes about 8,000 sq. miles of Upper or Barren Measures, 2,000 sq. miles of exposed Middle, and 12,420 sq. miles of exposed Lower Measures.

In a general way, the Missouri coal-field is bounded on the east by an imaginary line entering the northern part of the State just east of the 92d Meridian and passing across the State in a southwesterly direction to a point on the Kansas-Missouri line a little below the southern boundary of Barton county. An important part of the field overlaps this line and extends down the northern side of the Missouri river, underlying the greater part of Randolph, Howard, Boone and Audrain counties. In all, 34 counties produced coal during the year 1903. Fig. 1 shows the position of the coal-fields in Missouri.

Geological Features.

The Coal Measure formations, all of the Carboniferous age, rest unconformably upon the rocks of the sub-Carboniferous or Mississippian series, and, in some cases, upon the rocks of the Devonian and Silurian ages.

From their outcrop on the eastern and southeastern borders of the field, the Measures dip west to northwest at the rate of 10 ft. per mile, reaching a maximum thickness estimated by

* Read at the Supplementary Session of St. Louis, Mo.

[1] *Report of the Missouri Geological Survey*, 1894, p. 355.

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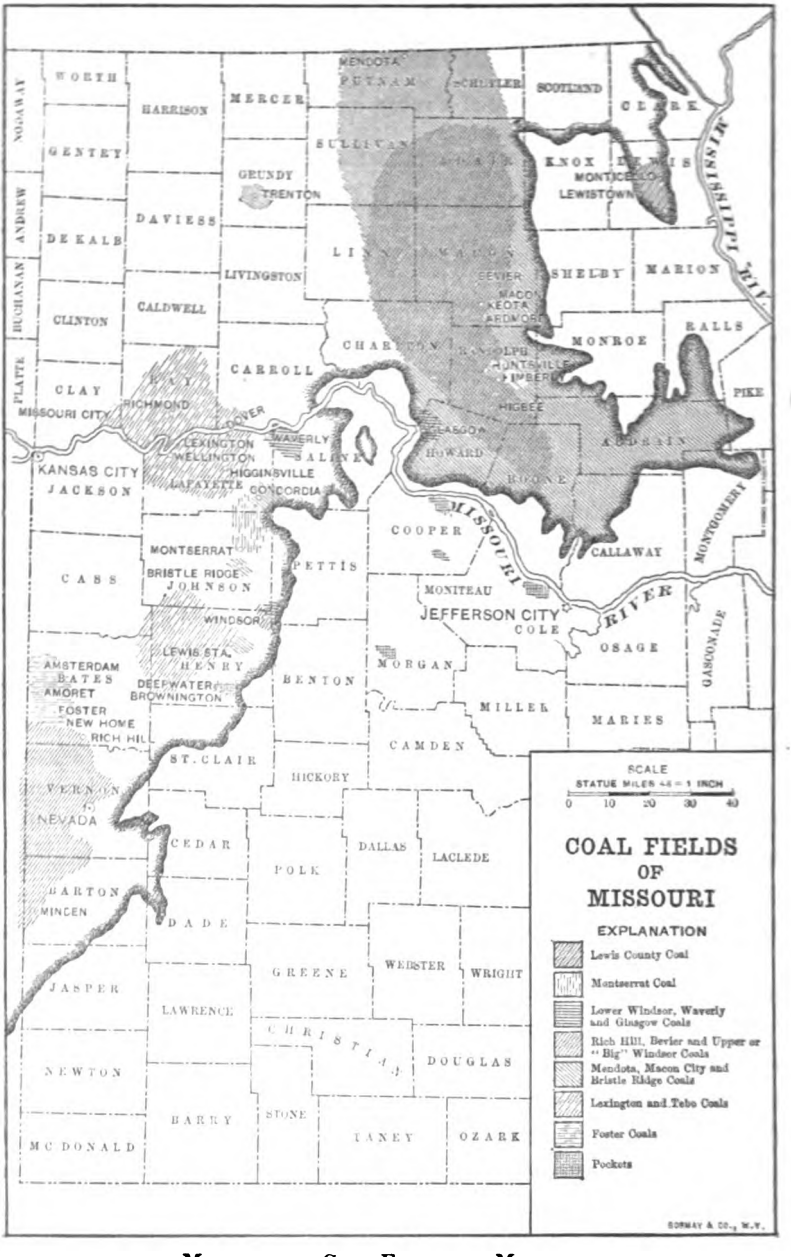
Fig. 1.



Map of the Coal-Fields of Missouri.

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Mr. Arthur Winslow[2] at 1,900 ft. A section, embracing the successive formations from the basal sandstone to the argillaceous limestone cap-rock of the Arkoe, Maitland and Quitman coal-horizons in Nodaway county, consists of interbedded limestones, sandstones, shales, fire-clays and the coals themselves, which in their fullest development are 29 in number. The coal-seams vary in thickness from 1 in. to 5 ft., often thinning out and disappearing entirely. The limestones are absent in the lower formations, but are quite prominent in the Upper Measures. The shales, in their various phases of argillaceous, bituminous, calcareous and arenaceous, are most prominent. The sandstones, although prominent in the marginal areas, are not prominent in the central portion of the field. The coal-beds are generally thin and variable in thickness and are sometimes represented by only a carbonaceous streak. The beds are generally underlain with clay and usually covered by black shales, or by a gray or drab clay-shale. For various reasons, the coal-formations have been divided into three classes, the Upper, the Middle and the Lower Coal-Measures.

Lower Coal-Measures.—The Lower Coal-Measures embrace eight coal-horizons resting on the basal sandstone and capped by the Mahoning sandstone. There is no limestone in these Measures. Two only of the eight horizons are sufficiently developed to be workable. The first is productive in a small way in portions of Lewis and Clark counties. The second horizon has yielded a considerable quantity of coal at Montserrat. With these two exceptions, these Measures are unproductive in this State, although all the coal from Arkansas and Indian Territory and the Big Muddy coal of Illinois, occur in the Lower Measures.

Middle Coal-Measures.—The Middle Coal-Measures embrace all the formations between the Mahoning sandstone and the Bethany Falls limestone. They contain 12 well-defined coal-horizons from which almost the entire output of the State is obtained. All the horizons, with the exception of the second, are capped by limestone rocks, generally of an argillaceous nature. All of the first four horizons are productive. From the first, the Windsor, Waverly and Glasgow coals are mined. In the second, the Rich Hill, Bevier and Windsor coals occur.

[2] *Report of the Missouri Geological Survey*, 1891, p. 23.

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The third yields the Mendota, the Macon City, the Brush Creek and the Bristle Ridge coals; from the fourth, the Lexington coal, so extensively mined throughout Ray, Lafayette and Clay counties is obtained. The eight remaining horizons are unproductive, with the exception of the tenth, which is mined in one locality in a small way.

Upper Coal-Measures.—The Upper or Barren Coal-Measures follow the Bethany Falls limestone and include nine coal-horizons, all of which are very thin and of little economic value, with the exception of the last, which has been mined a little in Nodaway and Atchison counties. The fifth has also been mined at times, but to a very limited extent.

The beds are generally regular and are associated with numerous shales, generally of the argillaceous and calcareous type, and with thick and persistent limestones, the Nodaway, St. Joe, Kansas City, Weston and Parkville limestones all occurring in these Measures. The Measures are capped by the Quitman limestone.

Geographical Features.

The coal-fields of Missouri are composed of various small beds, nowhere of any great extent and, in many cases, either too thin or occurring at too great a depth to be worked at a profit. The greater part of the coal mined to date has been taken from the marginal area where the beds are most numerous and lie nearest to the surface. These have to a great extent been exhausted and in the near future it will be necessary to look to the thinner and deeper beds. Under the present conditions, they cannot be worked profitably, and with such an abundant and cheap supply at hand on all sides, it is doubtful whether they ever will be mined.

Montserrat Coal.—The Montserrat coal underlies the greater portion of Johnson county and a portion of Lafayette county, and, at one time, was extensively mined near the town of Montserrat. The inferior quality of the coal, however, caused its abandonment some years ago, and at the present time it is only mined on a small scale. In thickness it runs as high as 5 ft., the lower 3 ft., however, containing almost all of the good coal. It slacks quickly and contains a large proportion of iron pyrite.

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Waverly Coal.—Waverly coal is found in the extreme north-eastern portion of Lafayette county, near the town of Waverly. The extent of this field is limited and of a very doubtful nature, while the bed is extremely difficult to approach, on account of the physical features of the country. The Coal-Measures, as near as can be learned from developments, show a workable vein, more or less faulted, about 4 ft. thick, with only one or two small bands of slate, which can be removed in mining. The coal rests on rock and has a good roof. A little gas is present and the room-and-pillar system of mining is needed to extract it. The coal is of a poor bituminous variety, low in fixed carbon and high in ash. A large percentage of sulphur in the form of pyrite also is present.

A coal of the same general characteristics is mined at Windsor and at Glasgow, which is believed to be the same bed. It is found also in the Bevier district, where it is known as the Eureka bed. It is excessively high in ash and is not mined in that field.

Bevier Coal-Field.—Bevier coal has the widest distribution of all the Missouri coals and from this bed the greater part of the State's tonnage is derived. It has been extensively mined for years at Huntsville, Ardmore and Bevier. It spreads out over Macon, Linn, Chariton, Randolph and parts of the adjoining counties. The coal varies in thickness from 4 to 6 ft. In the northern portion of the field it lies at an average depth of from 50 to 70 ft. from the surface, but deepens towards the south. The bed is cut into by the Moberly sandstone and also by the larger drainage-channel, and in many places is very difficult to approach. In the vicinity of Bevier the coal is mined by shafts. In the southern districts, however, advantage is taken of its numerous outcrops along the hillsides and the mines are principally drifts and slopes. The coal is mined by the room-and-pillar system. The roof is generally of a shaly character, grading at times into a sandstone. The sandstone affords a good roof, but the shale is more or less treacherous and requires considerable propping. The thickness of the seam is frequently diminished by horse-backs and rolls, which sometimes entirely cut out the coal.

The coal is mined extensively at Bevier, Huntsville, Higbee, Ardmore, Kimberly and Keoka. There are also many small

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mines that are worked intermittently. The coal itself is high in ash, moderate in moisture- and sulphur-content, and is the only coal that has successfully met the competition of the coals from neighboring States. The same coal is found at Rich Hill in Missouri, and at Leavenworth, Kansas.

Rich Hill Coal.—The Rich Hill coal-field lying in the southwestern portion of Missouri includes the western halves of Barton and Vernon counties, and the southern portion of Bates county. Near Rich Hill the coal has been found in numerous small basins or lenses of from 100 to 500 acres in area, closely adjoining each other. The coal varies in thickness from 2.5 to 5 ft. No coal has been found under Rich Hill, but there are two series of basins, one to the northwest and the other to the south of the town. The theory has been advanced that this coal was deposited in many small adjoining lakes.

The Rich Hill coal-field, although practically exhausted at the present time, was, for many years, the most productive one in the State and had few if any equals in the coal-fields of the West. The country around Rich Hill, for a radius of 15 miles, has been thoroughly prospected by the Rich Hill Coal Mining Co., but no coal was found to which it would be profitable to sink a shaft, with the sole exception of the New Home field, comprising a small basin of approximately 600 acres. This basin would be expensive to operate, owing to the irregularities in the undulating course of the coal-strata.

The principal operations in southern Missouri are at Minden, where the vein, which crops out just east of Minden, is the same as that so extensively mined throughout Crawford and Cherokee counties, Kan.

The bed averages about 42 in. thick and has a good roof of block-slate and a fire-clay bottom. The coal-seam has a uniform dip, frequently cut by clay-slips and horse-backs. Its quality ranks first in Missouri.

The Foster coal-bed is the only other one mined in this field. It averages about 30 in. thick, is very faulty and irregular and has a very treacherous roof. It is mined in small quantities near Worland, Amsterdam, Amoret and Foster. On account of the shallowness of the covering, most of the mining is done by stripping. The coal is low-grade, soft, fragile and slacks quickly on exposure to the atmosphere.

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The geological position of this bed has not been fully determined.

Mendota Coal-Field.—Mendota coal is the same vein that is extensively mined in southern Iowa. It underlies the eastern portion of Putnam county and the western portion of Adair and Schuyler counties, Mo., but its southern limit is not well defined. The bed, varying in thickness from 2 to 3.5 ft., is usually separated into three layers by pronounced clay- or shale-partings. The roof is generally of shale or slate overlain with 4 ft. of limestone. In many places, however, a soapstone has been deposited between the shale or slate, and the coal generally giving a roof that is bad. The bottom is a hard fire-clay. The region is hilly and the coal is frequently mined by drifting into the outcrops along the hill-sides. In other cases it is mined by the room-and-pillar system. The coal-bed is considerably cut up by faults and rolls. The coal lies at an average depth of 70 ft. below the higher country.

A bed, thought to be the same as the Mendota bed, is mined at Mexico in Audrain county, Macon City in Macon county, Brush Creek in Jackson county and near Concordia in Lafayette county. It is, however, too thin to be mined to any great extent. At Macon City the coal is of a very good quality and may yield a fairly good supply in the future.

Lexington Coal-Bed.—The Lexington coal-bed underlies the greater portion of Ray and Lafayette counties and a portion of Clay county. It lies from 200 to 300 ft. above the Waverly coal, and is found above the water-level at all points east of Lexington along the Missouri river. West of Wellington, it is found below the water-level. This vein, probably the first worked west of the Mississippi river, has been worked in the vicinity of Lexington for more than 75 years, and at one time a considerable quantity of coal was shipped from there by steamboats to places above and below on the Missouri river.

The opinion has been generally accepted that the entire Lafayette county was underlain by coal, but this is not true. The coal is entirely cut out by the Warrensburg sandstone, east of Lexington. This belt of sandstone reaches a maximum width of about 8 miles near the Missouri river, and tapers as it extends back towards Higginsville. Like a huge octopus, its many arms extend in every direction and cut out the coal on all sides.

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The bed is cut also by valleys of the large water-courses, and in many places the coal is entirely absent.

All the available territory fronting on the river around Lexington has been practically exhausted, although a small tonnage is still being mined west of the town.

There are no undeveloped coal-fields available at present to transportation-facilities, and those more remote are not sufficiently large to justify the expense of building railroads to reach them. At the present rate of consumption, profitable operations in this coal-field will be brought to a close in perhaps 15 years, although isolated openings may be subsequently operated.

The new mines of the Lexington Coal and Mining Co., south of Lexington, tap the largest and best field in the territory, which is continued for 5 or 6 miles southward of the mines and has an average width of from 3 to 4 miles. Several additional new mines could be operated in this field, all of which would prove profitable, but beyond that there is no field where mining can be carried on on a large scale.

The coal-seam varying in thickness from 16 to 24 in. has a clay-bottom, and is overlain by a block-slate which comes down with the coal. Above the slate is a solid blue limestone roof. The conditions are almost perfect for the long-wall system of mining.

The coal is of good quality and is well adapted to domestic and steam-purposes. A portion of the coal mined southwest of Higginsville, known as "farmers' coal," sells at a slightly higher rate than the regular Lexington coal. A peculiar feature of this coal is its saturation or impregnation with a thick, black, oily substance, resembling tar. The coal is very rich and pure and finds a ready sale for domestic use. For steam-purposes, however, it is no better than Lexington coal, and as the vein here is only 17 in. thick, it is more expensive to mine.

To the east of the barren belt formed by the Warrensburg sandstone, coal is again found near Dover, 11 miles east of Lexington. The field, originally small, has been further reduced by erosion. The coal is the Lexington seam with the same general characteristics, varying in some details, but not in a manner to lessen the cost of production. The quantity of

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the coal is good and equal to the best Lexington coal. The seam varies in thickness from 12 to 28 or 30 in., and is subject to rolls. The roof, though not so satisfactory as at Lexington, is not troublesome. The bottom is soft and the digging easy, the undercutting being done in a clay underlying the coal-seam. The costs of props offsets the easier mining and the increased thickness. This same coal is found in Ray county at the same relative level, and is operated on the same general plan. The same might be said of the coal in Clay county. The coal-seam dips very rapidly as the western boundaries of Ray and Lafayette counties are approached, and at Randolph Bluffs, 4 miles northeast of Kansas City, it lies at about 400 ft. below the surface.

Jordon Coal-Field.—The Jordan coal-bed is of very limited extent and is mined principally in and about the territory of Deepwater in Henry county. The coal is intercalated with beds of slate, and both roof and bottom are of shale. In many places the roof is poor, the water is troublesome, and the quality of coal not high. The vein, which is stated in the mine inspector's report to be 3 ft. thick, will be found considerably less when mined. The coal lies at an average depth of about 35 ft. below the surface.

Tebo Coal-Field.—Tebo coal is found chiefly in Henry county, with a small area overlapping into Bates and Johnson counties. This coal is also intercalated with slate and bands which have to be separated from it before the coal is ready for market. The vein has a good roof and varies in thickness from 20 to 30 in. The relation of this bed to the Lexington bed has not been fully determined.

Various Coal-Pockets.—Numerous irregular isolated deposits of coal are found in Missouri, usually just beyond the marginal area of the regular coal-beds. These irregular deposits are often of great thickness, as compared with the regular Coal Measures, having been found as thick as 75 feet. They are found at various depths, and, in many ways, they appear to be remnants of some former field that has been removed by erosion.

Geologists differ in their opinions regarding the origin of this coal. In many places it seems to be of a later formation than the regular beds. The coal is generally of poor quality,

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approaching lignite in character, and is often of the cannel type. Occasionally it also carries impurities in the form of iron pyrite, lead and zinc sulphides, and other kindred minerals. A curious occurrence of zinc blende is found with the coal in numerous places in Cooper, Morgan, Cole and Saline counties. I have heard of only one locality where a similar phenomenon has been observed. The zinc blende fills the larger vertical planes of the coal, and has evidently been deposited from solution.

These irregular coal-beds usually occur in the banks of streams and ravines and exhibit signs of disturbances. Although unconnected with one another, they bear the same general characteristics. They have never been mined to any great extent, but they have furnished a considerable quantity of coal for local use.

In connection with the theory that these irregular deposits are remnants of other fields, an unusual occurrence was noted at Lexington some years ago. Near the Lexington depot a shaft was sunk at a point where it was 100 ft. deep to the regular Lexington bed. Above the coal was a layer or two of thick limestone-rock. Above the rock were found drift, logs, lumps of coal weighing as much as from 300 to 400 lb., and scattered boulders. The whole rested in a bed of quicksand, showing that there had evidently been a disturbance, in which the regular Lexington coal had been torn up at some point and evidently re-deposited by the agency of water.

Methods of Mining the Coal.

Stripping.—Along the outcrop of the various beds, and in many other places where the physical conditions of the country permit, a considerable tonnage is produced from strippings. In several of the beds, from want of satisfactory roof-conditions, this is the only method by which the coal can be properly won.

In general, most of the work is carried on in the fall and winter, when the market-conditions are most favorable, and teams and labor are in least demand for other purposes. The coal is of inferior value and slacks quickly on exposure and handling. There were 47 pits in operation during 1903, in each of which more than 10 men were employed, and, in addition, many small pits were worked intermittently.

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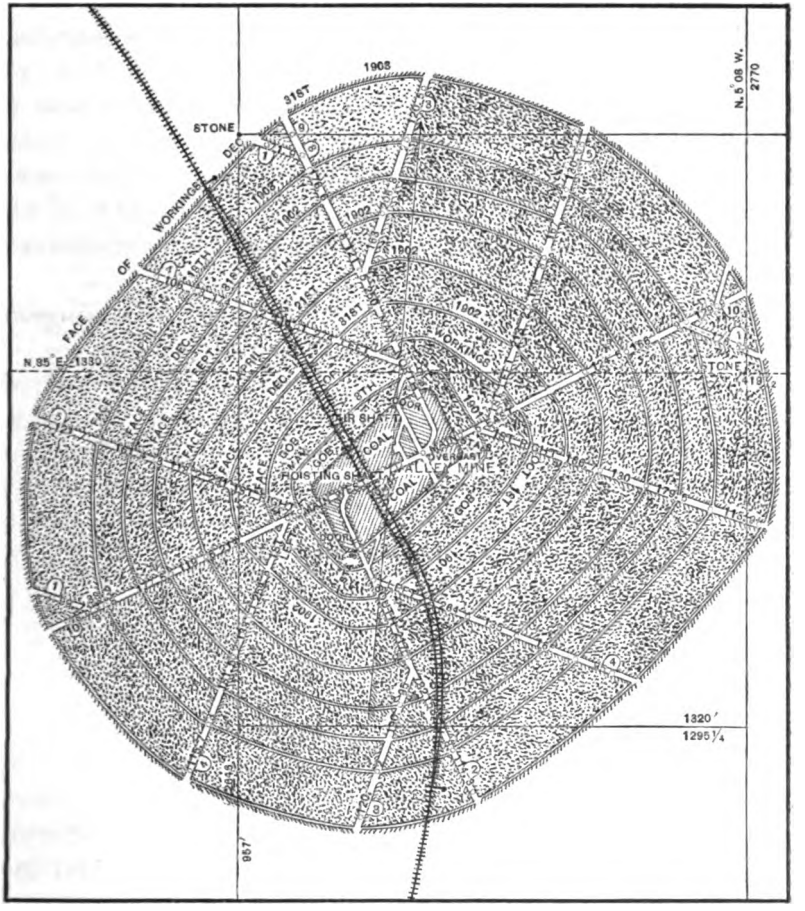
Room-and-Pillar.—This type of mining is the most common in Missouri, there being 164 mines so worked in 1903. In Missouri the shaft-pillars vary from 100 to 400 ft. square, depending on the nature of the roof, the overlaying strata and the depth at which the coal is mined. The entry-pillars are gener-

FIG. 2.



MAP OF ROOM-AND-PILLAR WORKING AND OPEN-PIT MINING AT MINDEN, BARTON COUNTY.

ally about 20 ft. wide and rooms are turned off at intervals of about 30 ft., leaving 10-ft. pillars between rooms. Cross-entries are turned about 300 ft. apart. The double-entry system is required by law with cross-cuts every 50 ft. Fig. 2 shows the method of room-and-pillar working and open-pit mining at Minden, in Barton county.

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The coal is generally shot off the solid, although in many places, especially in the Mendota field, the coal is undermined, cut on the sides and wedged down with a pick, no powder being used.

Long-wall.—The long-wall system, which offers the fullest advantages in mining the thinner seams of Missouri, is generally used in the thin seams. The Lexington coal-bed is extensively mined by this method, both in Lafayette and Ray counties, where the general thickness of the vein ranges from 16 to 24 in. By this system all of the coal is extracted in the best of condition. Ventilation is comparatively simple, and the haulage-distance for a given output is relatively short. Few roads are required to be kept open and the coal is easily worked. The cost of timber, track and other equipment is minimized and that of powder entirely saved.

The long-wall method of mining followed by the Lexington Coal Mining Co. at the Valley mine is shown in Fig. 3.

In order to understand properly the long-wall method of mining at the shaft of the Valley mine, Lexington, Mo., it is necessary to know something of the strata immediately above and below the coal. A section at the bottom of the shaft shows:

	Inches.
Limestone,	84
Roof-slate,	2
Block-slate,	6
Band-slate,	3
Coal,	20
Mining-dirt,	1
Sulphur-rock,	3
Fire-clay,	19
Limestone-bottom,	...

In opening up the Valley mine, two main entries were driven east and west from the shaft. At a distance of 150 ft. from the shaft, the main entry north was turned off the main east and the main entry south off the main west. At distances 75, and 125 ft. further, the branch-entries are also turned at an angle of 45 degrees. Cross-cuts were then driven between these entries, leaving a pillar to support the bottom, 200 ft. by 450 ft., in the shape of a parallelogram, the ends running north and south, but the sides so directed as to the bulk of the pillar around the main north and main south entries.

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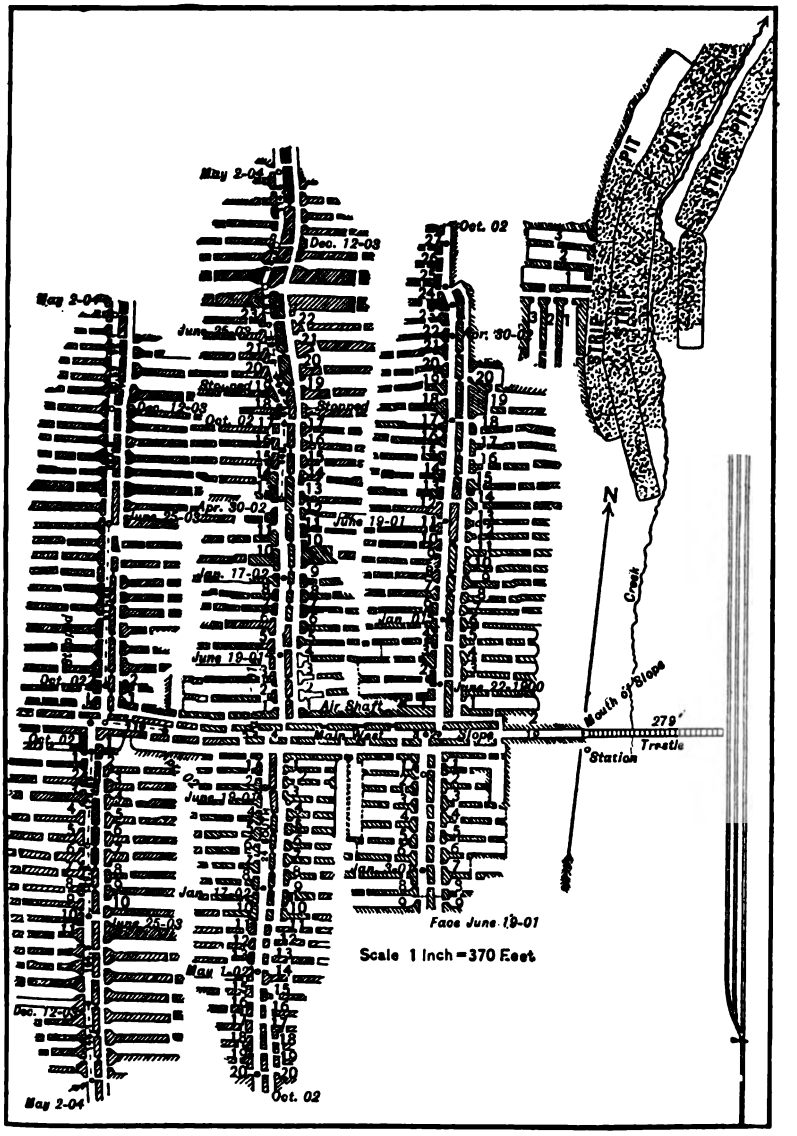
An air-course connects the air-shaft with the main north and main south entries, passing across the main east entry by means

FIG. 3.



MAP OF THE LONG-WALL WORKINGS AT THE VALLEY MINE, LEXINGTON, MO.

of an overcast which is tunneled above the limestone, or "head" rock as it is locally termed. The exterior ribs of the

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cross-cut form a long wall-face, which is worked in the following manner:

The fire-clay below the coal is removed by electric coal-cutters to a depth of from 2.5 to 3 ft., the cut being about 6 in. wide. The coal is spragged and held until the machine has passed 30 or 40 ft. The sprags are then removed and the gradual increase of roof-pressure upon the face breaks down the coal in large lumps. The two slates above and the sulphur-rock below come down with the coal, but separate from it in falling. Four machines are operated at the present time, making a continuous circuit around the face. They are operated night and day, and travel fast enough to mine each man's coal every other day. The miners spend the interval of time in loading the coal, packing and gobbing the walls with rock and slate, and in cleaning up their places.

The movable face-track resting on iron ties is pushed forward by each man, and an extra or fractional part of a rail-length inserted and the miner is again ready for the machine.

In this manner the face gradually advances and the roof settles behind it on the packing in the gob. The entries advance with the face and are kept open by gobbed walls and packings. When the main entries have advanced about 650 ft. from the last branch entry, a triangular gob-wall is built and another parallel cross-entry is turned. This arrangement confines the distance between entries at the face to about 450 feet.

On haulage-roads it is necessary to "brush" the roof to within 50 ft. of the face. The roof-slate and a portion of the overhead limestone are shot down and the fire-clay removed, giving an entry of about 6 ft. 8 in. Each miner is given a working-place of from 30 to 75 ft., according to ability. The distance between the packing and the face is about 7 ft. The hauling is done by mules. The mine is ventilated by a 12-ft. fan; 150 men are employed, producing 600 tons of coal per day, which is equivalent to 4 tons per man.

Statistics of Production.

The annual production of coal in Missouri during the past 20 years has fluctuated between 2,000,000 and 4,000,000 tons. The industry showed a steady growth up to 1888, reaching a tonnage that year of 3,909,967 tons. The following year the

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tonnage decreased more than 1,500,000 tons. In the period of five years following, a steady increase was maintained, and in 1893 the output was again increased to 3,000,000 tons. In 1894, however, it showed a decrease of 800,000 tons. Since 1894, the increase has been comparatively regular, and in 1902, the 4,000,000 mark was passed. During this period, the average price per ton at the mines has varied from $1.10 to $1.58 per ton, the former price being paid in 1897 and 1898, the latter price during 1903.

The lowest annual tonnage recorded for 20 years was in 1886, when the output amounted to only 1,800,000 tons. The highest production was reached in 1903, the record being 4,265,328 tons. In this year Missouri ranked fourteenth among the coal-producing States.

There were 348 corporations and individual companies mining coal in Missouri during 1903, of which one company produced one-fourth of the total output and ten companies considerably more than one-half. Eighty per cent. of the total quantity of coal mined in Missouri during 1903 was produced by nine counties as follows: Macon, 1,198,133; Lafayette, 539,612; Randolph, 450,181; Bates, 359,061; Adair, 312,403; Ray, 280,162; Vernon, 207,125; Barton, 200,433; Putnam, 125,543 tons. No other countries produced as much as 100,000 tons.

Character of Missouri Coals.

All of the coals of Missouri, with the exception of a few local deposits of cannel-coal, are of a distinctly bituminous nature, and are not of high grade. They are not adapted to the manufacture of gas on account of the large proportion of pyrites and ash present. They are comparatively soft and suffer quickly from exposure or excessive handling. They are essentially steam-coals and are mainly used for locomotives and household-purposes.

Note by the Secretary.—Comments or criticisms upon all papers, whether private corrections of typographical or other errors or communications for publication or "Discussions" or, independent papers on the same or a related subject, are earnestly invited.

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Proceedings of the Eighty-Seventh Meeting, Lake Superior, September, 1904.

COMMITTEES.

DULUTH.—Nelson P. Hulst, *Chairman;* J. B. Adams, W. C. Agnew, M. H. Alworth, C. W. Andrews, R. Angst, William R. Appleby, C. E. Bailey, G. G. Barnum, E. F. Bradt, Mylie Bunnell, George L. Cheesebrough, Peter Christensen, F. A. Cokefair, T. F. Cole, C. A. Congdon, N. Cowling, George H. Crosby, M. D. Cullum, C. d'Autremont, Jr., H. L. Dresser, C. T. Fairbairn, Walter Fitch, F. L. Gilbert, J. T. Hale, G. G. Hartley, E. B. Hawkins, M. S. Hawkins, J. H. Hearding, Louis Hill, L. J. Hopkins, F. E. House, J. C. Hunter, O. D. Kinney, J. W. Kreitter, E. J. Longyear, Alexander McDougal, W. A. McGonagle, A. M. Marshall, W. N. Merriam, T. D. Merrill, John Millan, P. Mitchell, Clarence Moore, Hon. Page Morris, C. H. Munger, Henry Nolte, W. J. Olcott, A. L. Ordean, Thomas Owens, F. A. Patrick, H. M. Peyton, D. M. Philbin, Capt. Charles Potter (U. S. A.), L. W. Powell, Harry Roberts, Louis Rouchleau, George A. St. Clair, J. U. Sebenius, Joseph Sellwood, Amos Shepherd, W. B. Silvey, George Spencer, George C. Stone, H. B. Sturtevant, George D. Swift, Kirby Thomas, C. D. Thompson, A. D. Thomson, G. A. Tomlinson, Charles Trezona, R. R. Trezona, Maj. J. H. Upham, Charles Van Barneveld, W. W. Walker, J. L. Washburn, A. C. Weiss, H. J. Wessinger, A. B. Wolvin, Dwight E. Woodbridge.

HOUGHTON.—W. E. Parnall, *Chairman;* W. D. Calverley, *Secretary;* W. B. Anderson, L. S. Austin, Jacob Baer, William Bath, F. J. Bawden, Simon Beahan, John Been, C. H. Benedict, Walter Bloomfield, S. H. Brady, Samuel Brady, Henry Brett, A. L. Burgan, F. I. Cairns, W. H. Cake, Arthur L. Carnahan, William Carpenter, Phillip Carroll, John J. Case, T. L. Chadbourne, W. A. Childs, B. F. Chynoweth, Captain James Chynoweth, H. P. Clausen, F. G. Coggin, H. D. Conant, J. B. Cooper, J. R. Cooper, John Cuddihy, Thomas Cummings, Robert Davidson, J. B. Dee, J. R. Dee, T. S. Dee, Theodore Dengler, M. M. Dennis, Captain Thomas Dennis, F. W. Denton, A. D. Edwards, R. M. Edwards, C. S. Fales, Daniel Fisher, Henry Fisher, James Fisher, A. L. Floeter, Frank H. Getchell, M. C. Getchell, W. M. Gibson, W. M. Gilliland, James G. Glanville, George Goodale, H. S. Goodell, A. R. Gray, Edward Grierson, Homer Guck, H. D. Haddock, Captain Josiah Hall, John L. Harris, Captain S. B. Harris, Enos Harrold, F. William Hartmann, George L. Heath, C. M. Hoar, Thomas Hoatson, O. P. Hood, Captain Thomas Hooper, Captain Joshua D. Hosking, Dr. L. L. Hubbard, Arno Jaehnig, E. D. Johnson, G. A. Koenig, Edward Koepel, C. H. Krause, R. H. Leach, A. W. Leonard, James MacNaughton, Edward McCormick, Thomas McDonald, R. T. McKeever, F. W. McNair, James McRae, John Mackey, John C. Mann, Major G. A. Marr, Captain Thomas Maslin, Harry Mercer, James Merton, Michael Messner, William F. Miller, Captain J. Milligan, Dr. J. W. Moore, Charles H. Moss, H. F. Nickerson, Charles L. Noetzel, P. H. Paine, John Penrose, Captain John Pentecost, Graham Pope, R. C. Pryor, James Ramsay, J. T. Reeder, Allen F. Rees, J. H. Rice, W. G. Rice, James

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Richards, Captain J. P. Richards, P. R. Roberts, James Robertson, W. H. Rowe, E. Ryan, C. W. Saunders, Dunbar D. Scott, A. E. Seaman, C. V. Seeber, R. R. Seeber, John Senter, R. Skiff Shelden, James W. Shields, John C. Shields, Robert H. Shields, Charles Smith, Fred. Smith, S. R. Smith, William Smith, Joseph Soddy, F. W. Sperr, F. McM. Stanton, Horace J. Stevens, Fred. Stoyle, Paul Swift, W. R. Thompson, W. J. Uren, Frank L. Van Orden, William Veale, Johnson Vivian, John M. Wagner, Edward Warne, P. O. Wels, Captain Thomas Whittle, Captain James M. Wilcox, H. E. Williams, Charles A. Wright, E. H. Wright, Z. W. Wright, M. F. Youngs, J. J. Zealand.

ISHPEMING.—W. H. Johnston, *Secretary;* John Q. Adams, O. C. Davidson, M. M. Duncan, Austin Farrell, T. A. Felch, Walter Fitch, W. F. Fitch, W. H. Green, John C. Greenway, H. R. Harris, N. M. Kaufman, S. R. Kaufman, W. B. Linsley, George R. Persons, E. W. MacPherran, A. Maitland, Samuel Mitchell, David T. Morgan, George A. Newett, James Russell, Joseph H. Vandeventer, John R. Van Evera, Thomas Walters, Peter White, Joseph H. Winter, H. O. Young, A. J. Yungbluth.

ST. LOUIS, MO.—Arthur Thacher, *Chairman;* F. E. Drake, *Treasurer;* H. A. Wheeler, *Secretary;* E. R. Buckley, B. F. Bush, Pierre Chouteau, Joseph A. Holmes, C. D. McLure, Hermann C. Meister, Phillip N. Moore, Russell Parker, William B. Potter, W. H. Reeves, John C. Robinson, R. A. B. Walsh.

The first session was held at the Assembly Room of the Commercial Club, Duluth, Minn., Wednesday, September 14, 1904, at 9.30 a.m.

President James Gayley called the meeting to order. A speech of welcome was made by the Mayor of Duluth, Dr. M. B. Cullom, followed by an address by Hon. Joseph B. Cotton, both felicitating the Institute on its visit to the Lake Superior region, and expressing the hearty pleasure and compliment that the people of Duluth felt in receiving the members. Mr. Gayley and Dr. Raymond responded in behalf of the Institute.

The Secretary announced the news, received by telegraph, of the death on September 12, 1904, of Mr. Philip W. Moen, a Vice-President of the Institute; and during the subsequent excursions news was received of the death on September 17, 1904, of Mr. Oliver Williams, of Catasauqua, Pa., who for many years was an active and beloved member of the Institute.

The following papers were presented by their authors:

Biographical Notice of Prof. William H. Pettee, by Dr. R. W. Raymond, New York, N. Y.

Centrifugal Ventilators, by R. V. Norris, Wilkes-Barre, Pa.

The following papers were presented in oral abstract by the Secretary in the absence of their authors:

Biographical Notice of Clement Le Neve Foster, by T. A. Rickard, New York, N. Y.

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The Case of Henry Cort, by Charles H. Morgan, Worcester, Mass.

The Genesis of the Diamond, by Gardner F. Williams, London, England.

The Secretary gave notice that, at the next annual meeting, an amendment to the Rules would be proposed, substituting for Rule IX. the following:

"The Council is hereby authorized to incorporate the Institute under the laws of the State of New York, and the Trustees of the corporation thus formed are authorized to adopt rules for the management of the Institute, and to execute such agreements as may be in their judgment necessary and advisable to enable the Institute to participate in the results of the generous offer of its member, Mr. Andrew Carnegie, to provide a building in the city of New York for the use of American engineering societies:"

The second session was held at the same place, Thursday, September 15, 1904, at 9.30 a.m., and was devoted largely to papers on blast-furnace practice.

The following paper was read, in the absence of the author, by Mr. F. S. Witherbee, Port Henry, N. Y.

Some Special Forms of Blast-Furnace Charging-Apparatus, by T. F. Witherbee, Durango, Mexico.

The following paper was presented in oral abstract by the Secretary, in the absence of the author:

Improvements in the Mechanical Charging of the Modern Blast-Furnace, by David Baker, Philadelphia, Pa.

These two papers were discussed together, the following gentlemen taking active part: Messrs. James Gayley, R. W. Raymond, B. F. Fackenthal, Jr., Julian Kennedy, D. T. Croxton, and Prof. B. Osann of Zollenfeld, Hartz.

The following paper was then read by the author:

A Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series, by C. K. Leith, Madison, Wis.

A short discussion followed the reading of the above paper, in which Messrs. T. F. Cole, Nelson P. Hulst and W. N. Merriam participated.

The Secretary read by title the following papers, and pre-

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sented them in printed form for distribution among those attending the session;

Appraisal of the Value of Minerel Lands with Especial Reference to Coal Lands, by Dr. H. M. Chance, Philadelphia, Pa.

The Concentration of Gold and Silver in Iron-Bottoms Reduced from Highly Ferruginous Copper-Mattes, with Notes upon the Metallography of Mattes, by Myrick N. Bolles, New York, N. Y.

The Commercial Wet Lead-Assay, by H. A. Guess, Silverton, Colo.

The Investigation of Alaska's Mineral Wealth, by Alfred H. Brooks, Washington, D. C.

Superficial Blackening and Discoloration of Rocks Especially in Desert Regions, by Prof. William P. Blake, Tucson, Ariz.

Report of a Committee to Co-Operate in Standardizing Abbreviations, Symbols, Punctuation, etc., in Technical Papers.

The Equipment of a Laboratory for a Smelting-Plant, by Herbert Haas, Ingot, Cal.

Evidences of Plication in the Rocks of Cananea, Sonora, by Prof. William P. Blake, Tucson, Ariz.

The Manufacture of Coke in Peru, by J. Morgan Clements, New York, N. Y.

Cyanide Practice at the Maitland Properties, South Dakota, by John Gross, Maitland, So. Dak.

Notes on the Flow of Gas from Orifices, by Prof. W. R. Crane, Lawrence, Kan.

Crushing in Cyanide Solution as Practiced in the Black Hills, South Dakota, by Prof. Charles H. Fulton, Rapid City, So. Dak.

A Geological Cross-Section through the Western Cordillera along the Rio Huasco, by S. H. Loram, Chile, S. A.

The Gold Mines of the District of San Pedro, Cerro de San Pedro, San Luis Potosi, Mexico, by George A. Laird, Sonora, Mexico.

Biographical Notice of Robert H. Thurston, by Dr. R. W. Raymond, New York, N. Y.

Testing Gold-Ores by Amalgamation, by Ernest A. Hersam, Berkeley, Cal.

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The Taviche Mining District near Ocotlan, State of Oaxaca, Mexico, by Dr. H. M. Chance, Philadelphia, Pa.

Geology of the Treadwell Ore-Deposits, Douglas Island, Alaska, by Arthur C. Spencer, Washington, D. C.

Discussion of paper of Mr. Edwards, "Concrete in Mining and Metallurgical Engineering," by F. T. Havard.

Discussion of Mr. Bolles' paper on "Concentration of Gold and Silver in Iron-Bottoms, Reduced from Highly Ferruginous Copper-Mattes, with Notes upon the Metallurgraphy of Mattes," by Edward Keller, Baltimore, Md.

Discussion of Prof. Blake's paper on "Superficial Blackening and Discoloration of Rocks, Especially in Desert Regions," by Prof. Theodore B. Comstock, Los Angeles, Cal.

The following papers and contributions, not named above, were read by title at the sessions, for subsequent publication and distribution:

Notes on the Occurence of Pebbles, Concretions and Conglomerate in Metalliferous Veins, by Edward Halse, Columbia, S. A.

The Constitution of Mattes Produced in Copper-Smelting, by Allan Gibb and R. C. Philp, Queensland, Australia.

The Genesis of the Copper-Deposits of Clifton-Morenci, Arizona, by Waldemar Lindgren, Washington, D. C.

The Decomposition and Formation of Zinc Sulphate by Heating and Roasting, by Dr. H. O. Hofman, Boston, Mass.

Casting-Machines for Pig-Iron, E. A. Uehling, New York, N. Y.

Moore Slimes Process, by George Moore, New York, N. Y.

Refractoriness of Some American Fire-Brick, by R. F. Weber, St. Louis, Mo.

Labor-Saving Appliances in the Commercial and Works Laboratory, by Edward Keller, Baltimore, Md.

The Roasting and Magnetic Concentration of a Blende-Marcasite Concentrate, by Dr. H. O. Hofman, Boston, Mass.

The Effect of Silver on the Chlorination and Bromination of Gold, by Dr. H. O. Hofman, Boston, Mass.

An Automatic Stock-Line Recorder for Blast-Furnaces, by J. E. Johnson, Jr., Longdale, Va.

Some Features of Blast-Furnace Action, by J. E. Johnson, Jr., Longdale, Va.

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Discussion of paper by Witherbee, "Some Special Forms of Blast-Furnace Charging Apparatus," by B. F. Fackenthal, Jr.

The third (supplementary and final) session was held in the Mines and Metallurgy Building, Block 74, St. Louis Exposition, Mo., on Saturday, September 24, 1904, at 2 p.m.

The meeting was called to order by Mr. Arthur Thatcher, Chairman of the Local Committee, who requested ex-Vice-President Robert H. Richards to preside. Mr. Richards called upon Prof. Joseph A. Holmes, who extended, in the name of the Governing Board of the Exposition, a hearty welcome to the members and friends of the Institute, to which Dr. R. W. Raymond responded in behalf of the Institute.

The following papers were presented in oral abstract by their authors:

The Lead Resources of Missouri, by Arthur Thatcher, St. Louis, Mo.

The Zinc-Smelting Industry of the Middle West, by Herman C. Meister, St. Louis, Mo.*

Ore-Dressing Practice in Missouri, by O. M. Bilharz, Flat River, Mo.

The Fire-Clays of Missouri, by H. A. Wheeler, St. Louis, Mo.*

The Joplin Zinc District of Missouri, by H. Foster Bain.†

The following papers were read by title by the Secretary:

The Coal-Fields of Missouri, by B. F. Bush, St. Louis, Mo.*

The Relation of Mining Engineering to Other Fields, by Robert H. Richards, Boston, Mass.§

Problems of American Mining Schools, by Samuel B. Christy, Berkeley, Cal.§

Mining Engineering and Mining Law, by James D. Hague, New York, N. Y.§

The Radium Exhibit of the U. S. Geological Survey at the St. Louis Exposition, by George F. Kunz, New York, N.Y.†

The following papers, first presented at the New York meet-

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* Printed copies of this paper were distributed at the session.

† Not furnished for publication.

* Printed copies of this paper were distributed at the session.

§ Read at the Mining Engineering Section of the International Congress of Arts and Science, September 24, 1904, and the title here entered by special permission of the Congress.

† Not furnished for publication.

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ing of the Iron and Steel Institute, October, 1904, were read by title for future publication, under a mutual agreement between the Councils of the two Institutes:

Acid Open-Hearth Manipulation, by Andrew McWilliam and William H. Hatfield, Sheffield, Eng.

Comparison of Methods for the Determination of Carbon and Phosphorus in Steel, by Baron H. Jüptner von Jarnstorff (Austria), Andrew A. Blair (United States), Gunnar Dillner (Sweden), and J. E. Stead (England).

The Influence of Carbon, Phosphorus and Manganese on the Tensile-Strength of Open-Hearth Steel, by H. H. Campbell, Steelton, Pa.

The Application of Dry-Air Blast to the Manufacture of Iron, by James Gayley, New York, N. Y.

The Development and Use of High-Speed Tool Steel, by J. M. Gledhill, Manchester, Eng.

Members and Associates Elected.

The following persons have been elected members or associates by postal ballot since the date of the last similar announcement in the Proceedings of the Atlantic City meeting of February, 1904:

(In response to Circular No. 1, January 29, 1904.)

Members.

Frank Vans Agnew, Assayer, ... London, England.
John J. Blow, Mining Engineer, ... Brooklyn, N. Y.
Moses Blumenkranz, Mining Engineer, ... Metcalf, Ariz.
John M. Boutwell, Mining Geologist, ... Washington, D. C.
R. M. Brock, Professor of Geology, ... Kingston, Ont., Canada.
Gelasio Caetani, Mining Engineer, ... Rome, Italy.
R. M. Cannon, Mining Engineer, ... Colorado Springs, Colo.
W. H. Case, Mine Manager, ... Tlalpujahua, Mexico.
Roscoe H. Channing, Jr., Manager of Mining Co., Salt Lake City, Utah.
Gerard A. Crane, Mining Engineer, ... London, England.
Burroughs Edsall, Mining Engineer, ... El Paso, Texas.
Arthur H. Elftman, Mining Engineer and Geologist, Minneapolis, Minn.
Edward L. Fuller, Coal and Salt Mining, ... Scranton, Pa.
Harvey G. Gilkerson, Civil and Mining Engineer, . Telluride, Colo.
Hubert J. Gould, Mining Engineer, ... Cobar, New South Wales.
J. T. Hawkins, Mine Superintendent, ... Anaconda, Colo.
Ellis W. Honeyman, Chemist, ... Morenci, Ariz.
J. August Ingols, Mining Engineer, ... Idaho Springs, Colo.
Edward S. Jones, Coal Operator, ... Scranton, Pa.

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Henry E. Jones, Mining Engineer, ... Bulawayo, So. Africa.
Rufus H. King, Superintendent of Coal Company, . Morgantown, W. Va.
Morris P. Kirk, Civil and Mining Engineer, . . . El Paso, Texas.
John O. Lampshire, Assayer, Chemist, ... Kofa, Ariz.
George Barstow Lee, Mining Engineer, Metallurgist, Bisbee, Ariz.
Jafel Lindeberg, President and Manager of Mining Co., Seattle, Wash.
Charles E. Locke, Mining Engineer, Metallurgist, . Boston, Mass.
William Longworth, Manager of Mining Co., . . Lithgow, New South Wales.
Jasper A. McCaskell, Mining Engineer, ... New York, N. Y.
George W. McNaughton, Manager of Mica Mines, Sydenham, Ont., Canada.
Randolph Meikleham, Electrical and Mining Engineer, New York, N. Y.
Harlan S. Miner, Chemist, ... Gloucester City, N. J.
Henry C. Mueller, Miner, ... Morenci, Ariz.
Edward E. Nelson, Mining, ... Guayaquil, So. America.
Masayuki Otagawa, Civil and Mining Engineer, . Tokio, Japan.
Juan de la Cruz Posada, Mining Engineer, Medelin, Colombia, So. America.
Richard Read, Mine Owner, ... Sydney, New South Wales.
Carl von Reischach, Mining and Metallurgical Engineer, San Francisco, Cal.
Paul Reisinger, Mining Engineer, ... Stockett, Mont.
August Roesler, President of Engraving Co., . . . New York, N. Y.
Alexander P. Rogers, Mining Engineer, ... New York, N. Y.
Charles M. Shannon, Mine Owner, ... Los Angeles, Cal.
Frank L. Smale, Mine Superintendent, ... Victor, Colo.
Harry B. Small, Mining Engineer, ... Ojuela, Mapimi, Mexico.
Edward A. Smith, Jr., Mining Engineer, ... Denver, Colo.
Elias A. C. Smith, Metallurgist, ... Canton, Baltimore, Md.
C. W. Van Law, Mining Engineer, ... Chicago, Ill.
Rudolph Weniger, Mining Engineer, ... Ojuela, Mexico.
Charles H. Wilson, Mine Manager, ... Brocko Creek, Australia.
Frederic E. Wright, Geologist, ... Houghton, Mich.
William H. Yeandle, Jr., Mining Engineer, . . . Placeres de Oro, Mexico.

Associates.

Charles C. Swift, ... Colorado Springs, Colo.
W. Irving Walker, ... Millington, Md.
Chojiro Yokogama, ... Cambridge, Mass.

Change of Status from Associate to Member.

William H. Landers, ... San Francisco, Cal.
A. Campbell McCallum, ... Victoria, B. C., Canada.
William G. Mather, ... Cleveland, Ohio.
Frank J. Nagel, ... Querida, Colo.

(In response to Circular No. 2, March 1, 1904.)

Members.

Clarence Eugene Abbott, Mining Engineer, . . . Eveleth, Minn.
H. Albert, Mining Engineer, ... New York, N. Y.
Percy P. Barbour, Mining Engineer and Surveyor, Idaho Springs, Colo.
Herbert Bailey Cox, Superintendent of Blast Furnaces, Cornwall, Pa.

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Gilbert Joseph Dawbarn, Lecturer in Engineering,
Ballarat School of Mines, South Australia.
Ernest Du Bois, Metallurgist, ... Mexico City, Mexico.
James W. Ellsworth, Coal Operator, ... New York, N. Y.
Lincoln Ellsworth, Coal Miner, ... New York, N. Y.
George Max Esterly, Mine Operator, ... Valdes, Alaska.
George Fergie, Mining Engineer, ... Copiapo, Chile.
Frank Herman Granstedt, Mine Manager, ... Sydney, New South Wales.
John Joseph Hamlyn, Mine Manager, ... Oroville, Cal.
Henry B. Hovland, Mining Engineer, ... Duluth, Minn.
Thomas Edward Johns, Mine Surveyor,
Feddeida Gold Mine, Transvaal, So. Africa.
Edward Horton Jones, Mine Surveyor, Metallurgist, Magdalena, Mexico.
Charles Durin Kaeding, Mining Superintendent, . Chemulpo, Korea.
Charles Joshua Kamper, Jr., Civil and Mining Engineer, Petros, Tenn.
William John Lakeland, Mining Engineer, Ballarat E., Victoria, Australia.
Henry Landes, Geologist, ... Seattle, Wash.
J. C. Henry Lubken, Consulting Engineer, . . . Johnstown, Pa.
J. King McLanahan, Jr., Manufacturer and Quarry Man, Hollidaysburg, Pa.
Stephen Cookman Miller, Mineralogist, Chemist, . Washington, D. C.
Edwin J. Polkinhorn, Assayer and Chemist, . . . Zacatecas, Mexico.
Oscar F. Scholz, Electrician, ... Pittsburg, Pa.
Clarence L. Shaw, Mine Superintendent, ... Ely, Nev.
Dr. William Frederick Smeeth, State Geologist, . Bangalore, India.
William Bryan Shropshire, Mining Engineer, . . Pittsburg, Pa.
Walter Techow, Ph.D., Mining Engineer, ... Bourne, Ore.
Thomas Kirby, Mine Manager, ... Superior, Wis.
Daniel Maurice Watters, Mining Engineer, . . . Sumpter, Ore.
Erastus Hutton Webster, Assayer and Chemist, . Durango, Mexico.
Aubrey Stuart Wheler, Mining Engineer, ... Johannesburg, So. Africa.
Robeson T. White, Metallurgical Engineer, . . . Bayonne, N. J.
John G. Witherbee, Chemist, ... Durango, Mexico.
N. B. Wittman, Iron Merchant, ... Pottstown, Pa.
Marshall William Wood, Mine Owner, ... Boise, Idaho.
Milford Wortham, Metallurgist, ... Pittsburg, Pa.
William Young, Mine Operator, ... Rat Portage, Ont., Canada.

Associates.

Theodore Albert Burbidge, Student, 111 West 103d Street, New York, N. Y.
Elbert H. Gary, U. S. Steel Corporation, 71 Broadway, New York, N. Y.

Change of Status from Associate to Member.

Harry L. Williams, Mine Manager, ... Bristol, Tenn.

(In response to Circular No. 3, April 18, 1904.)

Members.

Axel E. Anderson, Draughtsman, Surveyor, . . . Denver, Colo.
Morris Baker, Mechanical Engineer, ... Philadelphia, Pa.
Edwin L. Beck, Mill Man, ... Kendall, Mont.

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Herbert Powers Bowen, Mining Engineer, ... Cos. Sonora, Mexico.
Thomas Ellis Brown, Consulting Engineer, . . . New York, N. Y.
Arthur Harry Collbran, Mining Engineer, ... Seoul, Korea.
Tenney C. De Sollar, Mining, ... Golden, Colo.
Robert L. Edwards, Mining Engineer, ... Ulysses, Idaho.
Edgerton E. Fillmore, Mining Engineer, ... Kay-Moor, W. Va.
Nile O. S. Ford, Mining Engineer, ... New York, N. Y.
Alexander Fraser, Metallurgist, ... Boulder, West. Australia.
R. B. Gage, Mine Superintendent, ... Niagara, Ont., Canada.
Alvah W. Gifford, Mine Manager, ... El Paso, Texas.
Wilmer D. Glenn, Mining Engineer, ... Joplin, Mo.
Stephen L. Goodale, Assayer, ... Colorado Springs, Colo.
Richard L. Grider, Mining Engineer, ... Seiad Valley, Cal.
Forbes William Guernsey, Mining Engineer, . . Trail, B. C.
Edwin Giles Hart, Mine Superintendent, ... Ocoltan, Oaxaca, Mexico.
Max Heberlein, Consulting Metallurgist, ... New York, N. Y.
Charles M. Hicks, Mine Superintendent, ... Austinville, Va.
Max R. Hirschberg, Mine Manager, ... Teller, Alaska.
A. E. Johnson, Mill Superintendent, ... Colorado Springs, Colo.
Charles Henry Jones, Chemist, ... Santiago de Cuba, Cuba.
Eugene Patrick Kennedy, Mine Superintendent, . Susanville, Ore.
Halstead Lindsley, Mining Engineer, ... Telluride, Colo.
John Stuart McKaig, Mining Engineer, ... Dewey, Idaho.
Frederick W. Mathews, Mining Engineer, ... East Rand, So. Africa.
William Maxwell, Mill Owner, ... Georgetown, Australia.
James V. B. Murdoch, Mining Engineer, ... Chihuahua, Mexico.
Henry T. Murray, Chemist, ... Clifton, Ariz.
James Brown Neale, President of Coal Company, . Minersville, Pa.
W. B. Pearson, Mine Foreman, ... Frisco, Utah.
Roscoe L. Peterson, Mining and Mechanical Engineer, New York, N. Y.
Peter Quartano, Assayer and Surveyor, ... Zeehan, Tasmania.
Etienne A. Ritter, Mining Engineer, ... Colorado Springs, Colo.
John Seward, Mining Engineer, ... Orange, N. J.
Walter Sydney Sharwood, Mine Surveyor, ... Randfontein, Transvaal.
Jo E. Sheridan, Mine Inspector, ... Silver City, New Mexico.
Frank Sloat, Assistant Mine Superintendent, . . . Daiquiri, Cuba.
Lawrence D. Somers, Assayer and Surveyor, . . . St. George, Utah.
Isidore Tom, Mining Engineer, ... London, England.
Frank H. Trego, Mining Engineer, ... Platteville, Wis.
Charles A. Vaux, Mining Engineer, ... Lakefield, England.
Harry B. Wallis, Mining Engineer and Metallurgist,
Graylands, Horsham, Sussex, England.
J. O. Willard, Civil Engineer, ... Pittsburg, Pa.

ASSOCIATES.

Hugh J. Carney, ... Ouray, Colo.
Walter A. Emeis, ... Golden, Colo.
Albert C. Franck, ... Kansas City, Mo.
Robert P. Franck, ... Kansas City, Mo.
Ferdinand J. Fohs, Geologist, ... Marion, Ky.
Frank C. Hill, ... Carthage, Mo.

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Charles Holt, ... New York, N. Y.
William D. Kilbourn, ... Pueblo, Colo.
Lionel Lindsay, ... Berkeley, Cal.
Henry P. Nagel, ... Denver, Colo.
Scott H. Sherman, ... Denver, Colo.
W. Irving Spencer, ... Golden, Colo.
Loyal W. Trumbull, ... Golden, Colo.
George J. Wackenhut, ... Colorado Springs, Colo.
Oscar M. Weichsel, ... New York, N. Y.

Change of Status from Associate to Member.

E. Davenport Cleland, Mine Manager, ... Coolgardie, Western Australia.
R. Prewitt Coleman, Mining Engineer, ... Encampment, Wyoming.
Roger T. Pelton, Mining Engineer, ... Bisbee, Ariz.

(In response to Circular No. 4, June 10, 1904.)

Members.

John Fordyce Balfour, Mining Engineer, ... Sekondi, West Africa.
Henry C. Beeler, Mining Engineer, ... Cheyenne, Wyoming.
Arthur J. Boynton, Assistant Manager Blast Furnace, Elyria, Ohio.
James Bullen, Mine Manager, ... Pontywain, England.
Cecil A. Burne, Mining Engineer, ... Las Esperanzas, Mexico.
Richard T. Dana, Civil and Consulting Engineer, . New York, N. Y.
James Mitchell Daniel, Jr., Mining Engineer, . . Aguascalientes, Mexico.
Frank G. Drum, Manager Mining Estate, ... San Francisco, Cal.
Edmund W. Dwight, Treasurer Coal Company, . Philadelphia, Pa.
Charles E. Eagan, Mining Engineer, Metallurgist, New York, N. Y.
Charles H. Early, Mine Manager, Bulawayo, Rhodesia, So. Africa.
Edward H. Emerson, Mining Engineer, ... Colombia, So. America.
Hugh B. Fergusson, Mine Surveyor, ... Bullion, B. C., Canada.
David L. H. Forbes, Mine Superintendent, . . . Metcalf, Ariz.
Elias S. Gatch, Manager of Mining Company, . . St. Louis, Mo.
S. J. Hallett, Mine Manager, ... Aspen, Colo.
Norman Follett Harriman, Chemist and Engineer, Omaha, Neb.
Austin Hebbard, Chemist and Metallurgist, . . . Johannesburg, So. Africa.
Harry Peters Henderson, Mining Engineer, . . . Butler, Nev.
Herman Victor Hesse, Mining Engineer, ... Fairmont, W. Va.
James Edwin Hyslop, Mine Manager, ... Parral, Mexico.
George D. James, Mining Engineer, ... Denver, Colo.
Charles Edmund Kenney, Mining Engineer, Chemist, Angels Camp, Cal.
Clarence P. Linville, Metallurgist, ... State College, Pa.
George Alan More, Mine Surveyor, ... Beaconsfield, Tasmania.
Edward C. Musgrave, Mining Engineer, Mine Superintendent,
Mount Sicker, Vancouver Island.
John W. Prout, Jr., Instructor in Chemistry, . . Tucson, Ariz.
George S. Riviére, Metallurgist, ... Chicago, Ill.
Alfred Mayer Rock, Geologist, ... Flat River, Mo.
George Rose, Cyanide Chemist, ... Octave, Ariz.
John Sanders, Metallurgist, ... San Fernando, Mexico.

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Irving H. Sevier, Foreman of Cyanide Plant, . . Kofa, Ariz.
Adam T. Shurick, Mining Engineer, ... Las Esperanzas, Mexico.
Baird Snyder, Jr., Superintendent of Coal Company, Lansford, Pa.
Vernon V. Sparks, Mining Engineer, Chemist, . . Baker City, Ore.
Harry Patterson Stow, Mine Owner and Manager, . Forbestown, Cal.
Otto Sussmann, Mining Engineer, ... New York, N. Y.
Samuel W. Vaughen, Manager of Blast Furnace, . Lorain, Ohio.
Samuel Whitworth, Engineer, ... Kalgoorlie, Western Australia.
Edward N. Zern, Mining and Mineralogy, ... State College, Pa.

Associates.

Marc Bailey, Student, ... Houghton, Mich.
Frank A Dalburg, Student, ... State College, Pa.
William G. Demarest, Student, ... New York, N. Y.
Ray L. Herrick, Student, ... Lehigh University, Pa.
Hansaburo Hunter, Student, ... London, England.

Change of Status from Associate to Member.

Sidney M. Bamberger, ... Salt Lake City, Utah.
Harry J. Wolf, ... Telluride, Colo.

(In response to Circular No. 5, August 16, 1904.)

Members.

Charles I. Auer, Assayer and Chemist, ... Durango, Mexico.
Rafael Flores Alatorre, Mine Manager, ... Hidalgo, Mexico.
Francis K. Baxter, Jr., Chemist, ... Ingot, Cal.
Charles Lazarus Bryden, Assistant in Chemistry, . Easton, Pa.
Douglas Bunting, Mining Engineer, ... Wilkes-Barre, Pa.
Edward W. Clark, Mine Superintendent, ... Ophir, Utah.
Frederick D. Dimmick, President of Mining Company, Birmingham, Ala.
Alexander George Dixon, Mining Engineer, . . . San Luis Potosi, Mexico.
William James Dooley, Mine Owner and Operator, Salt Lake City, Utah.
Frederick D. Fuller, Metallurgist, ... Sumpter, Ore.
George E. Gunn, Mining Engineer, ... Salt Lake City, Utah.
Lee Hager, Geologist, ... Houston, Texas.
Walter Raleigh Hamilton, Mining Engineer, . . Bodie, Cal.
Irving Herr, Mining Engineer and Metallurgist, . Guanajuato, Mexico.
Herbert L. Holt, Mining Engineer, ... Edgemont, Nev.
Byron Nicholson Jackson, Assayer and Chemist, . Bodie, Cal.
Robert Rollin Jones, Metallurgist, ... South Chicago, Ill.
David Lewis Killen, Mine Operator, ... Sumpter, Ore.
Harry Kipp, Chemist, ... Chilpancingo, Mexico.
Glenn VanBuren Levings, Mining Engineer, . . . Chihuahua, Mexico.
Carl Paul Ludwig, Consulting Mining Engineer, . Birmingham, Ala.
Charles Willis Macdougall, Mining Engineer, . . Hancock, Mich.
C. H. Munger, Mine Manager, ... Duluth, Minn.
John Jermain Porter, Chemist, ... DuBois, Pa.
Andrew Bryson Rawn, Civil and Mining Engineer, Dott, W. Va.
John A. Redfern, Mine Inspector, ... Hibbing, Minn.

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Arthur F. Rising, Mining Engineer, ... Madoc, Ont., Canada.
John Burgoyne Stewart, Mining Engineer, ... El Oro, Mexico.
A. A. Talmage, Civil Engineer, ... Los Angeles, Cal.

Change of Status from Associate to Member.

James Carson Agnew, Mining Engineer, ... Hibbing, Minn.
Sidney M. Bamberger, Mining Engineer, ... Salt Lake City, Utah.
W. Arthur Sloan, Mining Engineer and Metallurgist, Silver City, New Mexico.
George O. Smith, Assayer and Chemist, Melbourne, Victoria, Australia.
George K. Taggert, Chemist, ... Brooklyn, N. Y.
Loyal W. Trumbull, Professor Mining and Geology, Laramie, Wyo.
H. T. Willis, Mining Engineer, ... Champaign, Ill.
Harry J. Wolf, Mining Engineer, ... Telluride, Colo.

Excursions and Entertainments.*

Steamer-Trip on the Great Lakes.

The Lake Superior Meeting of the Institute was prefaced by a delightful voyage on the Great Lakes from Buffalo to Duluth on the splendid steamer *North West* of the Northern Steamship Co., which made an extra trip as a special accommodation for the party.

The steamer left Buffalo on Saturday evening, September 10, 1904, having on board nearly a hundred members and guests of the Institute, which number was subsequently augmented by others who joined the party en route.

The passage through Lake Erie was made without mishap, fears of rough weather on the comparatively shallow waters of this lake fortunately not being realized. On arrival at Cleveland Sunday morning, those of the party who were on deck had the opportunity of witnessing the method of loading coal direct from car to freight-steamer. A loaded car was run by gravity into a cage at the foot of a hoist and directly opposite an open hatchway of the steamer. The car was then gripped firmly in place and the cage with the enclosed car was elevated a sufficient height and turned over, dumping the coal into a wide iron hopper, and thence through an appended chute direct to the hold of the vessel. The foreign contingent of the party —our "Deutscher Verein"—was especially impressed with this rapid method of loading coal.

Sunday, September 11th, was bright and clear, and a religious service was held on the elevated sun-deck of the steamer.

* This account was prepared at my request by Dr. Joseph Struthers.—R. W. R.

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The run through the Detroit river in the afternoon was very interesting and enjoyable. The beautifully clear river, winding in and out like a ribbon between the green shores, together with the many small islands which dotted its surface, presented a pleasing and memorable picture. A short stop was made in the afternoon at Detroit, and later the beauties of the famous park at Belle Isle were intensified by heavy thunder-clouds in the west, which rendered the verdure even deeper and more attractive in color. The sail through the St. Clair river was full of interest. The many little streams and channels encircling the numerous islands of various sizes which were often connected by attractive rustic bridges—the picturesque houses of both home and club—the yachts and small craft dotting the surface of the water—all justified the name of the "American Venice" which has been applied to this beautiful stretch of land and water. The entrance into Lake Huron was made in the evening, and the early risers of the party who were on deck the following morning were rewarded for their good habits by a vista of magnificent cloud-pictures which were spread in glorious profusion along the horizon.

Mackinac Island was reached before noon on Monday, September 12th. At this place the clearness of the water, together with the beautiful cloud-effects in the sky, made an appropriate setting for the picturesque scenery of the rocky and precipitous shores.

The run through the lakes and the lower St. Mary's river was memorable for the large number of eastward-bound vessels which were passed en route, and, in viewing them, it was not hard to realize that the tonnage which passes through this river during the short open season of water-transportation exceeds in quantity that carried through the Suez Canal during the entire year. The *North West* arrived at Sault Ste. Marie, familiarly known as the "Soo," too late to permit individual parties to "shoot" the rapids of the St. Mary's river in small boats (a famous attraction of the Soo); but the disappointment to the younger members of the party was soon replaced in part by the interest attending the passage of the steamer through the immense Government locks, which are among the largest in the world.

Tuesday, September 13th, opened with threatening skies, and a little rain fell during the run through the Portage river.

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The concluding portion of the steamer-trip was passed without special incident, and the entire outing was unanimously regarded as one of continuous pleasure and benefit. The cool bracing air and the general prevalence of clear weather made the sun-deck a place of rare enjoyment.

Those who desired to study in advance the professional papers printed for the sessions at Duluth, were provided with copies; forty-one papers (aggregating about pages) being available at all times during the voyage.

Duluth, Minn.

Duluth, "the Zenith city," was reached early on Tuesday evening, and, through the activity of the local committee, the party was soon transferred by carriages to the Spalding Hotel, where old acquaintanceships were renewed with the members and guests of the Institute who joined the party there, and new ones made.

An interesting feature of the Duluth sessions was the unfurling of the "Institute Flag," which had been sent by the "color-bearer," Mr. J. Harry Lee, of Baltimore, Md., to grace the occasion.

On Wednesday, September 14, 1904, after the first session, and a substantial luncheon, members and guests were taken on the yacht *Vidette* and several launches, to inspect the U. S. Government work at the south or Superior entry to the harbor. The pier under construction is the largest piece of monolithic cement-work of sub-aqueous character yet undertaken. Practically all of the foundation is laid under water at depths varying to 17 ft. or more, without the aid of a single diver, the work being controlled entirely from the surface.

In the evening a reception and collation were given by the Northland Golf Club, at the club house, where a brilliant company, comprising the leading citizens and ladies of Duluth, entertained the visitors.

On Thursday afternoon, September 15th, the steamer and launches were again called upon to carry the party to the works of the Zenith Furnace Co., where an informal, but none the less enjoyable, luncheon was served *al fresco*. The plant of the company consists of a blast-furnace, 76 ft. 6 in. high, with bosh of 16 ft. 9 in., and hearth of 10 ft. diameter. The blast is

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heated in three Gordon stoves 60 ft. high and 20 ft. in diameter. The capacity of the furnace is 250 tons of pig-iron daily, and the slag produced is granulated and sluiced to the adjacent water front where it is utilized to fill in docking-space.

In addition to the blast-furnace plant, the company has 50 Otto-Hoffman by-product coke-ovens, treating 400 tons of coal daily, to yield 300 tons of coke. The average output of by-products amounts to 4,000 gal. of tar, 10,000 lb. of concentrated ammonia (20 per cent. of NH_3), and 1,250,000 cu. ft. of illuminating-gas of 18 candle-power. The company has also an auxiliary carburetted water-gas plant of 2,000,000 cu. ft. daily capacity, in order to supply gas when the blast-furnace is idle. The cities of Duluth and Superior have contracted for the daily product of oven-gas, produced by the company, for a term of ten years.

The docks of the company are fully equipped for handling and storing a large tonnage of coal. The complete dock is 1,200 ft. long, 350 ft. wide, having a slip 150 ft. wide, dredged 3,000 ft. to a depth of 21 feet.

The party returned by water to Duluth in time to board the special trains which were ready for the trip to the iron-ore ranges.

Train-Trip on the Iron-Ore Ranges.

The excellent and complete account of the trip to the Minnesota and Mesabi ranges, written by Mr. Dwight E. Woodbridge, correspondent of the *Iron Age*, and published in the columns of that paper September 22, 1904, is here adopted with slight changes, principally for condensation:

Two trains were utilized for the trip on the ranges, one a full "Pioneer Limited" service of the Chicago, Milwaukee & St. Paul Railroad, and the other, made up of private and official cars from Duluth. Friday morning, September 16th, the party spent at Ely, inspecting the Chandler, Pioneer, Zenith, Sibley and Savoy mines of the United States Steel Corporation. These mines are located in the oldest rocks of North America, the "Ely Greenstones," in a canoe-shaped basin, about 2 miles long and about 0.25 mile wide at its greatest width. The strike is approximately northeast by east, the pitch at the east end steeply toward the west, and at the west end, perhaps as sharply, toward the east. The dip is nearly vertical. The

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deepest shafts slightly exceed 1,000 ft., and the deepest drill-holes are about 1,300 feet deep. Along the course of this basin the Oliver Iron Mining Co. has erected three shaft-houses of steel, one of which is 168 ft. high, the tallest in the world, and the others are but a few feet lower. These have been built of this great height on account of the necessity of grading the ore for size, and passing it over grizzlies between the skip-pocket and the car-loading pocket below. A number of the party, including many ladies, went underground at the Pioneer mine, where the broken and friable hard ores are mined by new adaptations of the caving system. These properties are yielding about 2,000,000 tons a year of Lake Superior ores of the best grade.

In the afternoon the great Mesabi range was reached, and the Biwabik and Duluth mines were first visited. Both are open pits, one loading with steam-shovel in the sides of an immense pit, 2,500 ft. long and 1,000 ft. wide, in the form of an immense horseshoe, with a surface over a large part of the area fully 75 ft. thick. Four shovels are used in the mine, but only for a small part of the time. This number is needed on account of the suddenness of change in the chemical quality of the ore throughout the pit and the necessity of getting proper ores to fill guarantees. The Duluth mine also is an open pit, but its ore is trammed to a shaft and hoisted, as in the underground system, although it was formerly a milled mine. The Biwabik mine has produced 925,000 tons a season, and during 1904 it will ship about 500,000 tons. The Duluth mine produces regularly about 150,000 tons a year. Next were visited the Fayal and Adams mines at Eveleth, which are combinations of underground and open-pit methods. The Fayal mine has two great pits, one for the steam-shovel and one for milling, or tramming, to a skip. It is working on a new pit, where 2,000,000 yd. of material are being moved from a steam-shovel mine. It is now mining about 30,000 tons a week, a very low rate of production. The Adams mine during 1903 mined from underground 1,600,000 tons, which is the world's biggest record for a shaft-mine. It is now developing a milling-pit, where six mills were in operation on the second level. This was the first glimpse that the party had had of a milling prop-

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erty and it was very much enjoyed. From the Adams mine the trains ran to the Mountain Iron mine, where about 1,000,000 tons a year are being moved from an open pit. This mine is now opened a mile north and south, and its greatest width is about 1,200 feet. Shovels are now working in the fourth ore-level. The product of this mine is apparently limited only by the possibilities of the railway that serves it. It is the only large open-pit mine that is entirely held under fee, so that its operators pay no royalty, and the thickness of stripping, as compared to the depth of ore beneath it, is so small that mining, including the stripping-charge, is done at an astonishingly low figure. From the Iron Mountain mine the party went to Hibbing, where the night was spent. During the evening an entertainment was held in the Hibbing Hall, and various members of a customarily staid and quiet party instructed and amused their associates by the most remarkable and delightful "stunts." A feature of the evening was the singing of German mining songs by the German chorus of nine voices.

During the entertainment at Hibbing, our "Deutscher Verein" guests,—Messrs. Theo. Beckert, Breslau; Prof. B. Osann, Zellerfeld im Harz; Fr. Kœnig and Max Ludwig, Eisleben; Hans P. W. Menzel and R. Gœtze, Bochum, and H. D. Klein, Dahlbruch, Westfalen; Z. F. Baum and W. Frenzel, Essen—expressed their thanks for courtesies received in a short and appropriate address, which was delivered by Mr. Beckert. [*Secretary's Note.*—The international character of the company was acceptably completed by the genial presence of M. Jules Goujon, the well-known ironmaster of Moscow—a Frenchman from Russia, thus representing two nations at once.—R. W. R.]

Saturday morning, September 17th, the trains pulled into the Stevenson mine, from which about 1,500,000 tons of ore will be moved during 1904. This mine is an enormous V-shaped body of high-grade coarse Bessemer ore which is in great demand by furnacemen. Three steam-shovels are working in ore producing up to 15,000 tons daily. The pit is about 0.5 mile, east and west, in ore, and is being widened by new stripping

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on the north bank. Later in the morning the party went into the Mahoning mine, a vast oval opening, arranged with standard-gauge tracks in spirals, to the third ore-level. A Sunday quiet prevails at the Mahoning constantly, and the mine can produce an amount of ore far in excess of its present output, which averages about 1,000,000 tons a year. Later the Hull and Rust underground mines of the Oliver Mining Co. were seen. Neither mine is now active, but the company is now figuring on removing the surface and making them the largest open-pit mines in the world. This work will probably be undertaken soon, though no announcement of a decision has been made. The company's Burt pool mine was next visited, which produces by the open-pit steam-shovel method about 1,000,000 tons of ore a year. The mine is arranged so that its mining-costs are about as low as those of any property on the Mesabi range. After visiting the Clark, Monroe-Tenor and other properties, the party returned to Duluth, inspecting on arrival the docks of the Duluth, Missabe & Northern road, which are now handling about 950,000 tons of ore a month.

The immensity of the Lake Superior iron-ore deposits may be inferred from the appended summary of production classified by ranges.

	Tons (2,240 lb.).
Marquette range, from 1855 to 1903 (49 years),	69,746,409
Menominee range, from 1877 to 1903 (27 years),	45,988,855
Gogebic range, from 1884 to 1903 (20 years),	40,731,186
Vermillion range, from 1884 to 1903 (20 years),	20,738,205
Mesabi range, from 1892 to 1903 (12 years),	66,640,349

Classified by 10-year periods the production since 1854 has been :—

10-*Year Periods.*	Tons (2,240 lb.).
1854–1863 inclusive,	620,127
1864–1873 inclusive,	6,169,941
1874–1883 inclusive,	15,596,912
1884–1893 inclusive,	56,881,841
1894–1903 inclusive,	164,358,874
Total,	243,627,695

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The proportions of different grades of iron-ore, shipped from the mines in 1903, were:—

	Per Cent.
So-called magnetic ores, consisting partly of magnetite and partly of hematite,	1.3
Hematite ores, hard specular, limonites and soft red hematites,	97.7
Soft red hematites and limonitic ores,	95.6
Hard ores—specular and magnetic and specular,	4.4

The limonitic ores are largely the intermixed ocherous ores of the Mesabi range. On the Marquette range the Beaufort, Titan, and some other mines have produced this variety of ore.

The record of loading iron-ore by steam-shovel at several mines during a 10-hour shift, was:—

	Burt Mine.	Mountain Iron Mine.	Mahoning Mine.	Stevenson Mine.
Date,	Aug. 6, '03	Aug. 13, '04	1902	July 28, '04
Make of shovel,	Marion 91	Marion 91	Bucyrus	Marion 98
Weight of shovel,	93 tons	93 tons	65 tons	100 tons
Capacity of dipper,	2½ yd.	2½ yd.	1¾ yd.	3 yd.
Number cars loaded,	152	162	.....	201
Gross tons loaded,	5,096	4,826	4,100	7,109

NOTE.—The weight of 1 cubic yard of iron-ore is equivalent to about 2 tons.

An excellent photograph of the party on the flat cars used for conveyance into the open pit of the Mahoning mine was taken while literally "standing on iron-ore." A half-tone representation of this interesting view is given on page 202, and on page 203 is given a list of the names of all the members grouped, for ease of identification, in four rows to correspond to the arrangement in the photograph.

A unique social feature of the train-trip was the organization of the members occupying the "stag" car, Ojibway, into a pseudo band of Indians bearing the name of that well-known tribe; and, as a mark of progress in civilization, it is recorded that during the return run from the Mesabi range to Duluth, all the ladies of the party (and none of their escorts) were corralled by telegraph in the club car and entertained at an Indian pow-wow trimmed with light refreshments. This circumstance shows the uplifting influence of American femi-

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ninity. Whether the tribe's action was due to the admiration of the irresistible charms of the fair sex, or to the recognition of their superiority in the "pow-wow" line, is a debatable question; but, judging from the happy expression on all faces (exclusive of the neglected escorts), the affair was an unquestioned success.

An interesting and valuable pamphlet of 30 pages and one map, giving an outline of the program of the sessions and excursions at Duluth, was prepared by the Local Committee and distributed among the members and guests. A brief description of the iron-ore ranges of Minnesota was also included, which is not here reproduced for the reason that Mr. Leith's paper, read at the second session at Duluth,* treats this subject in greater detail.

The pamphlet also includes a short description of the railroads traversing the ranges, together with data pertaining to their transportation-facilities of rolling-stock and docks, and quantities of ore annually shipped over them. Tables are also given showing the average analyses (Fe, P, SiO_2, Mn, Al_2O_3, CaO, MgO, S, moisture and ignition-loss) of cargo-shipments of Lake Superior iron-ores during the season of 1903, together with detailed annual shipments of iron-ore from all the mines on the different ranges up to and including the year 1903. These shipments aggregate the almost incomprehensible total of 243,627,695 long tons. These tables are published in the annual U. S. Geological Survey reports on the Production of Iron-Ores, by Mr. John Birkinbine, which are presented gratuitously on application to the Washington office of the Survey; for this reason, together with the exigency of limited space, they are not here repeated.

The increase in the capacity of ore-carrying vessels on the Great Lakes is shown by the subjoined table of vessels that have passed through the "Soo;" and even larger cargoes have been carried from Escanaba to ports in Lake Michigan.

* *Trans.*, xxxv. ——.

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TOP ROW (standing on the flat-cars). Read from left to right.

H. J. Wolff, Telluride, Colo., Dr. Joseph Struthers, New York City, N. Y., R. V. Norris, Wilkes-Barre, Pa., J. B. Adams, Duluth, B. J. Vallatt, Virginia, Minn., Ward Ames, Sr., Duluth, E. F. Remer, Stevenson, Minn., J. F. Myrick, San Francisco, C. M. Myrick, San Francisco, Julian Kennedy, Pittsburg, F. E. House, Duluth, William Wilkins, Ashland, Wis., G. L. Collord, Clairton, Pa., H. C. Farnum, Detroit, F. A. Eustis, Boston, J. M. Sherrerd, Easton, Pa., A. T. Shurick, Las Esperanzas, Mexico, J. A. Capp, Schenectady, N. Y., J. M. Clements, New York, N. Y., Prof. Howard Eckfeldt, So. Bethlehem, Pa., Dr. H. Ries, Ithaca, N. Y., Fried Kœnig, Eisleben, Germany, George J. Krebs, Somerset, Pa., L. W. Powell, Duluth, T. M. Eynon, Philadelphia, De B. Whitaker, Daiquiri, Cuba, Miss M. Ormrod, Allentown, Pa., S. H. Pitkin, Akron, Ohio, G. W. Pfeiffer, Daiquiri, Cuba, Olof Wenstrom, Boston, George Mesta, Pittsburg, C. A. Congdon, Duluth, A. H. Viele, Duluth, L. J. Merritt, Duluth, W. Frentzel, Essen, Germany.

SECOND ROW (seated on the settees of the cars).

Jawood Lukens, Conshohocken, Pa., Mrs. G. B. Richards, Buffalo, N. Y., Master Theodore D. Richards, Buffalo, N. Y., A. J. Rossi, New York City, Mrs. Julian Kennedy, Pittsburg, Mrs. George D. Swift, Duluth, T. F. Cole, Duluth, Mrs. L. Holbrook, New York City, Amos Shepherd, Hibbing, Levi Holbrook. New York City, E. S. Hutchinson, Newtown, Pa., Miss M. Wood, Philadelphia, Howard Wood, Philadelphia, F. Z. Schellenberg, Pittsburg, Mrs. J. Lukens, Conshohocken, Pa., Mrs. R. W. Raymond, Brooklyn, N. Y., Mrs. K. Thomas, Superior, Wis., Mrs. A. H. Viele, Duluth, Mrs. C. Whitaker, Allentown, Pa., James Gayley, New York City, Mrs. B. F. Fackenthal, Jr., Easton, Pa., Miss Raymond, Brooklyn, N. Y., Mrs. H. B. Kummel, Trenton, N. J., Miss A. A. Mesta, Pittsburg, Mrs. H. B. Sturtevant, Duluth.

THIRD ROW (seated on front edge of the cars).

W. G. LaRue, Baraboo, Wis., J. B. Guinn, Joplin, Mo., G. A. Voskule, Wynberg, South Africa, W. A. McGonagle, Duluth, Miss Mills, New York City, J. F. Lutes, Biwabik, James C. Hunter, Duluth, Mrs. H. W. Craver, Pittsburg, Nelson P. Hulst, Duluth, D. T. Croxton, Cleveland, H. W. Craver, Pittsburg, A. H. Eustis, Boston, G. E. Abbott, Eveleth, Minn., Ralph Gillette, Minneapolis, Dr. R. W. Raymond, Brooklyn, N. Y., Professor Osann, Clausthal, Germany, Miss Rachel Wood, Philadelphia, Theodore Dwight, New York City, B. F. Fackenthal, Jr., Easton, Pa., A. R. Bellenger, Syracuse, N. Y., Dr. H. B. Kummel, Trenton, N. J., S. W. Croxton, Cleveland, W. A. Barrows, Jr., Sharpsville, Pa.

BOTTOM ROW (standing in front of the cars).

F. W. Getty, Chicago, W. C. Agnew, Duluth, Jules Goujon, Moscow, Russia, Walter Wood, Philadelphia, Thomas Owens, Duluth, Paul S. King, New York City, C. W. Kieswetter, Duluth, H. H. McNamara, Duluth, George D. Swift, Duluth, Harry Thomas, Hibbing, G. G. Hartley, Duluth, John Uno Sebenius, Duluth, Joseph B. Cotton, Duluth, R. R. Trezona, Eveleth, Hans Mentzel, Bochum, Germany, W. J. Olcott, Duluth, George Bartol, Cleveland, Theodore Beckert, Breslau, Germany, Max Ludwig, Eisleben, Germany, H. L. Dresser, Duluth, R. F. Gœtze, Bochum, Germany, Kirby Thomas, Superior, Wis., D. B. Meacham, Cincinnati, F. S. Witherbee, New York City, W. N. Merriam, Duluth, H. B. Sturtevant, Duluth, C. H. Munger, Duluth, J. Lodge, Tuscaloosa, Ala.

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Increase in Capacity of Vessels on the Great Lakes Since 1856.

5-Year Periods. Ending.	Name of Vessel of Greatest Capacity.	Tons.	5-Year Periods. Ending.	Name of Vessel of Greatest Capacity.	Tons.
1856.	Schr. Wyandotte.	400	1886.	Str. Onoko.	2,500
1861.	Schr. Courtright.	460	1891.	Str. E. C. Pope.	2,800
1866.	Bark J. P. March.	697	1896.	Barge Aurania.	5,223
1871.	Schr. Negaunee.	1,148	1901.	Barge Manila.	7,400
1876.	Schr. J. Martin.	1,360	1904.	Str. A. B. Wolvin.	10,300
1881.	Str. J. N. Glidden.	1,484			

On account of the general interest, the following data, relating to the loading and unloading of iron-ore from freight-steamers on the Lakes, have been reproduced from the pamphlet above mentioned.

Ore-Shipments from Docks of the D., M. & N. R.R.

Year.	Average Tonnage.	Time Loading. Total.		Time Loading. Per 1,000 Tons.	
		Hours.	Minutes.	Hours.	Minutes.
1895.....	1,809	4	00	2	13
1896.....	2,214	5	16	2	24
1897.....	3,541	7	40	2	10
1898.....	3,550	8	10	2	7
1899.....	3,803	7	24	1	57
1900.....	3,783	6	31	1	43
1901.....	4,459	5	57	1	20
1902.....	4,814	7	9	1	29
1903.....	5,668	8	47	1	33

The maximum daily shipment of iron-ore from the docks of the D. M. & N. R.R., 65,758 tons, was loaded on August 31, 1903, and the shortest recorded time of loading was 5,250 tons in 30.5 min. (equivalent practically to 175 tons per min.), which was loaded on the steamer *J. H. Hoyt* in the year 1903.

The following record of loading the steamer, *Augustus B. Wolvin*, August 5, 1904, at the Great Northern Docks, Allouez, is of interest.

Boat arrived at dock,	2.45 p.m.
Started to load,	4.16 p.m.
Finished loading (about 9,000 tons),	4.50 p.m.
Shifted to complete cargo,	45 min.
Loading completed (10,245 tons),	5.45 p.m.
Total time at dock,	3 hr.
Total time loading,	1.29

Time of unloading at Conneaut 4 hr. and 30 min.

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Sunday morning, September 18th, was clear, cool and bracing. A large number of the party, including many of the Ojibway Indians, attended divine service at the First Presbyterian Church in order to hear the second interesting public address of Dr. Raymond, which was scientific, broadly cultured and altogether delightful. In the afternoon carriages were provided for a trip around the city and its environs, notably the celebrated boulevard drive, which is laid out on an ancient shore-line terrace of Lake Superior, 500 ft. above the present level of the lake. Some of the members drove to the docks of the Zenith Furnace Co. and inspected the largest ore-carrying steamer in the world, the *Augustus B. Wolvin*, of 10,300 tons capacity. A tasteful collation, provided through the courtesy of Mr. Augustus B. Wolvin, was served in the commodious officers-room of the ship.

Houghton, Mich.

Monday, September 19, 1904, found the two special trains of the Institute at Houghton, Mich., where a two days' stay was made in the copper-region. Representatives of the local committee were early on hand to extend a cordial welcome, and the party was soon transferred to a special train which was provided for the visit to the many mines and furnaces in the vicinity.

The thorough and comprehensive account of the visit to the copper-country is here republished from the *Iron Trade Review* of September 29, 1904.

"Two days were passed among the copper-region mines, from the Wolverine on the north to the Champion on the south, taking in Tamarack, Calumet, Atlantic, Champion and Baltic. Only a very brief summary of points of most interest can be given. Wolverine was inspected underground, and astonishment was expressed by those fortunate enough to make the trip at the wonderful way in which the hanging-wall held up. The lode here pitched at an angle of about 40°, and for a width of nearly 1,000 ft., and a height on the pitch of from 400 to 500, the lode is mined out with a very few 30-ft. pillars and an occasional floor varying from 2 or 3 ft. to 6, to hold it all from coming down. The shafts are sunk without timber and with only 30-ft. walls on either side. The mine uses no timbering underground, the openings are not back-filled and it

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does not cave. Wolverine is making about 8,000 lb. copper per month, and at the rate of 30 lb. to the ton of rock treated. This is as high as the best work of any amygdaloid mine in the region and is next only to the conglomerate-lode of the Calumet and Hecla.

"Tamarack's magnificent surface equipment at No. 5 shaft, which is 4,938 ft. deep, consists of a 6,500-h.p. Nordberg four-cylinder engine operating a double conical drum, the ends of which are 16 ft. in diameter and the center 24 ft. This drum runs two 6-ton skips in balance, each with 6,500 ft. of 1.5-in cable, and the hoisting is done at the rate of 3,000 ft. per minute. This shaft has a full duplicate hoisting-plant. It was sunk in 6 years on the assurance from the manager of the company that he would cut the lode at a certain depth and in a certain time. The work was actually done within eleven days of the time set, and the lode was found 12 ft. deeper than anticipated. The lode is the Calumet conglomerate and its pitch is almost perfectly true.

"Surface machine-plants at various Calumet plants showed a bewildering array of engines and compressors of almost every type, from the walking-beam and an 8,000-h.p. quadruple hoist, at the Red Jacket, hoisting 10-ton loads 4,000 ft. a minute, to the latest Nordberg or an Allis 1,500-h.p. twin vertical-tandem compound-expansion direct-connected operating a 1,090-kw. alternating-current generator of the General Electric type. Some Calumet machinery is a fearful and wonderful sight. The property has about 70,000 h.p., including hoists, pumps and generators, most of them in duplicate. There are 130,000,000-gal. daily pumping-capacity at the mills, for the purpose of handling sands in and below the stamps, and for hoisting waste-sands to launders. Practically 6,000 tons of tailings are run off into Lake Linden daily, and this is elevated by a 64-ft. sand-wheel to launders. The company has three of these wheels, one of 40, one of 50 and the newest of 64 ft. diameter. Calumet mills contain 28 stamps, of an average capacity of over 500 tons of rock daily. This conglomerate rock is easier to stamp than the harder amygdaloid of the other Lake mines. It is supposed to carry about 50 lb. refined copper to the ton. One of these mills is of steel construction, the other is to be, and when rebuilt, the two will treat about 10,000

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tons of rock per day, or more than the three largest gold-mines of the world combined. Calumet owns about 1,200 dwellings, which are leased to operatives at enough to pay a low rate of interest, and the town contains about 40,000 people, most of whom are supported more or less directly by this mine.

"Champion and Baltic are new mines, the first opening in each having been made about 5 years ago. They are making more than 1,000,000 lb. of copper monthly each, which is about half the copper the present plants are capable of. They are steadily increasing. Both are opened on the so-called Baltic lode, which is the oldest amygdaloid in the region and carries a high percentage of copper. Both mines are splendidly equipped with machinery especially adapted for their requirements and good for hoisting from 1,500 ft. and lower. Baltic mill is perhaps the most improved in the region with the exception of the new Osceola; and was an interesting spectacle. Osceola's mill, by the way, has one steeple compound stamp that is crushing 700 tons of hard amygdaloid per 24 hr. and has made a record of 725 tons daily for a week. This stamp is showing an economy of 30 per cent. in steam over single cylinders in the same mill. Champion is making openings at the rate of about 7,000 ft. annually and has now, since 1899, more than 6.5 miles of workings, with four shafts averaging more than 1,000 ft. depth. It is running on very rich rock and is splendidly opened. Atlantic was visited. It is the leanest mine in the world, its rock averaging a value, at 12.5 cents a pound for refined lake copper, or about $1.70 a ton. From this it is making money and will pay a dividend this year. But the visitors were treated to a surprise by the presentation of specimens of rock out of its ashbed-lode that averaged 10 per cent. of copper. Atlantic and Baltic use a dam on Salmon Trout river that impounds 1,250,000,000 gal. of water and saves pumping to the mills, averting a cost of at least 2 cents per ton of rock. This is a steel gravity-dam, of interesting design, and is surmounted by a trestle of the Copper Range railroad.

"At the Michigan Smelting Works, built by the Stanton and Paine interests and designed to serve their mines, the especially interesting thing was the mechanical casting of refined copper from furnaces, which is in direct opposition to the hand-meth-

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ods employed at Calumet, and the combination of hand and mechanism at Osceola. The question of oxidation of molten copper during the pour is being fought over by advocates of the two methods. This works is the largest in the district with an ultimate capacity for 60,000,000 lb. of copper and contains the biggest reverberatory furnaces ever built, each with capacity to treat 90 tons of ore a day."

During the time spent at Houghton and vicinity the local committee kept the members and guests visiting the numerous mines, mills and furnaces at such a rapid rate that the capacity to study and digest mentally the many interesting and instructive things was sorely taxed.

The Houghton Local Committee distributed an attractive pamphlet of 20 pages and one map, which gave the itinerary of excursions for Monday and Tuesday, Septemper 19 and 20, 1904, together with a brief description, by Horace J. Stevens, of the chief geological features of the copper-country, and the principal active mines in the district. This portion of the pamphlet was so valuable for an outline reference that it is herewith reprinted.

General Information of the Geology and Mines of the Lake Superior Copper-District.

BY HORACE J. STEVENS.

The map, Fig. 1, gives an outline of the developed portion of the Lake Superior copper-district. The Keweenawan formation is very extensive, outcropping on Michipicoten Island, north of Sault Ste. Marie on the eastward, and running through northern Wisconsin into northeastern Minnesota on the west. Attempts at mining have been made at various points along this stretch of about 400 miles, but the developed and developing mines of the present day extend for a distance of some 70 miles only, from the Phœnix mine on the northeast to the Victoria mine on the southwest.

The Keweenawan trap-belt is a syncline, the southern edge of which carries the developed mines. The syncline dips under Lake Superior at various angles, ranging from 22° with the horizon at the Arnold mine, Keweenaw county, to 73° at the Baltic mine, Houghton county. The trap-belt consists of a

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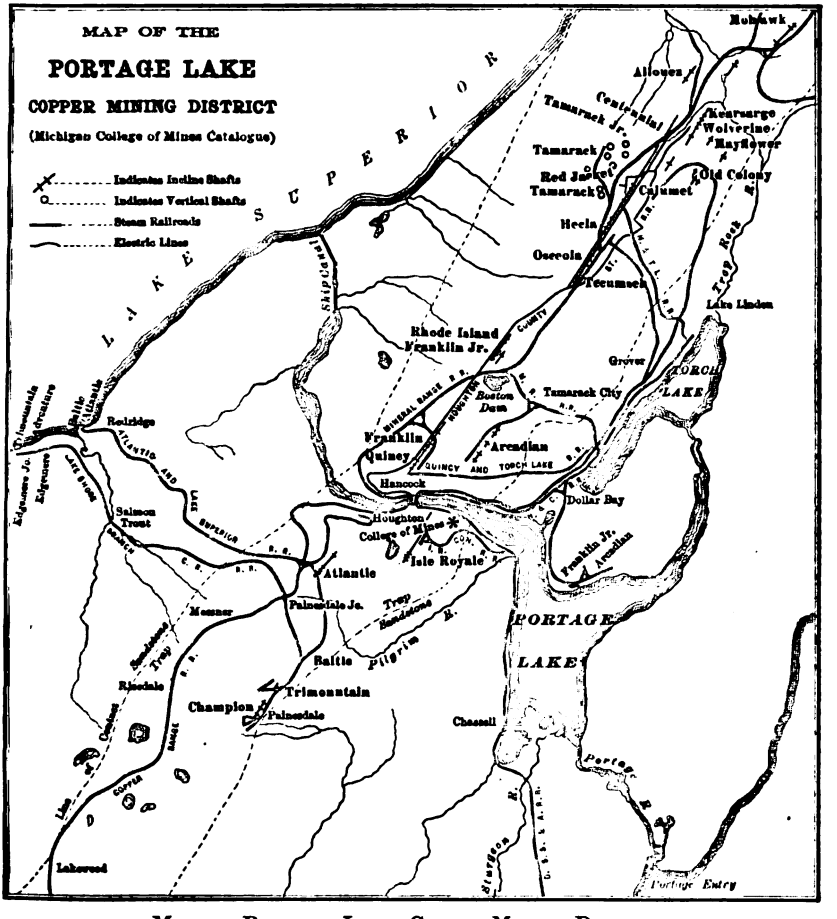
series of flows, perhaps nearly 200 in number, with intercalary beds ot conglomerates, perhaps 20 or more in number. The amygdaloidal portions of the trap-flows frequently carry native copper, though not all of them are workable. There is also much local variation from point to point in the copper-values

Fig. 1.



Map of Portage Lake Copper-Mining District.

found in these beds, locally termed amygdaloids. Many of the amygdaloids have undergone such extensive mineralological changes that they have lost all semblance of their original character, and carry epidote, prehnite, calcite, laumontite, etc., in large quantities. The copper occurs native in various forms in the amygdaloids. Occasionally immense masses, many tons

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in weight, are found; while in the richer mines there is much "barrel work," *i.e.*, masses of virgin metal, ranging in weight from a few pounds up to a 100 lb. or more.

The principal reliance of the mines is upon stamp-rock, that is, amygdaloidal or conglomerate-rock carrying native metal in sizes ranging from that of a pea down to flakes of almost microscopic dimensions that are saved with difficulty from the slimes at the mills.

The conglomerates are of aqueous or igneous origin, and perhaps of both. Where cupriferous, the conglomerates are impregnated with copper that has replaced the original cementing material; and the Calumet conglomerate, the richest bed of the district, often shows small masses of metal, up to several inches in diameter, that have completely replaced pebbles and small boulders. Copper has been found in a number of the conglomerates, but is being mined from two of them only. The Calumet conglomerate is mined by the Calumet & Hecla and also by the Tamarack, the latter working the underlay by means of vertical shafts. The Calumet conglomerate was found unprofitable both north and south of the Calumet & Hecla lands. The Allouez, or Boston & Albany, conglomerate is worked at the Franklin Junior mine. In addition to the bedded deposits there are fissure-veins, usually crossing the strata at approximately right-angles, with an approach to vertical dip. The fissures were much worked in early days and gave several mines that paid millions of dollars each in profits. At present the only active fissure-vein mine is the Phœnix in Keweenaw county. This mine produced the largest mass of native copper ever found, weighing 527 net tons; and the present production is largely heavy copper, though there is a fair proportion of stamp-rock. With the development of the industry and the increase in depth attained, the importance of masses has decreased, and the main reliance of all mines working the amygdaloid and conglomerative-strata is upon stamp-rock, at the present time. The heavy copper is found, as a rule, in the upper workings of the mines, though continuing downward for perhaps 3,000 ft. in the richer mines. Below a depth of 4,000 ft. a decrease in metal-values is noted in every mine that has exceeded that depth. None of the mines opened on fissure-veins have reached a depth of 3,000 ft., the copper-values in

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the stratified beds proving more persistent at depth. The deepest shaft of the district, and of the world, is No. 5 Tamarack, a vertical shaft bottomed at a depth of 4,938 ft. from the surface. In addition to the native-copper deposits, there are occasional fissure-veins, carrying copper-ores, but none of these have proven of permanent value, the mineralization being confined, as a rule, to a comparatively short distance each side of the crossing of a copper-bearing stratified bed. In addition to the more common copper-ores, such as chalcopyrite, bornite and chalcocite, several arsenical ores peculiar to the district have been identified, among these being the interesting compound arsenides of copper and nickel, known as mohawkite and keweenawite, found in a fissure-vein at the Mohawk mine, Keweenaw county. Owing to the peculiar geological and mineralogical conditions of the district, a single mining company may open several entirely distinct mines on a single tract. Among such instances is the Calumet & Hecla, which, in addition to its main mine on the Calumet conglomerate, has a parallel mine, 730 ft. eastward, opened on the Osceola amygdaloid, while a third parallel mine is now being opened on the Kearsarge amygdaloid, about one-half mile eastward, and search is being made for a supposed conglomerate nearly a mile to the westward of the main mine. The Tamarack also works the Osceola as well as the Calumet lode, opening both beds by vertical shafts, so that it may be said that separate mines are opened and worked by the same company, either in tandem or in teams. The Centennial mine made a failure of the northern extension of the Calumet conglomerate, then moved eastward on its own lands and opened a second mine on the Osceola amygdaloid, which, not promising well, was closed, and a third mine was opened on the Kearsarge amygdaloid. This latter began production two months ago, and gives promise of making a rich property. The Allouez company is also opening its third parallel mine, and its shares are selling at the rate of $1,250,000 for the property, though the shaft has not yet reached the Kearsarge lode. So regular are the master lodes of the district that there is no question whatever that the Allouez will cut the bed desired, and the time and depth are foretold within a few days and a few feet. The following description gives brief detailed references of the principal active mines

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of the Lake Superior copper-district, located in the three counties of Houghton, Keweenaw and Ontanagon.

Adventure.—The Adventure mine is at Greenland, Ontonagon county. Samuel Brady, superintendent; Roy Dunstan, mill superintendent. Company was organized in 1898 and consolidated the old Adventure, Hilton and Knowlton mines. Former was opened in 1850 along a line of old pits, dug by prehistoric miners. Property is still in the development stage, though stamping a small quantity of rock. Produced 2,182,608 lb. of refined copper in 1903.

Ahmeek.—The Ahmeek mine is at Kearsarge, Keweenaw county. William E. Parnall, superintendent. The company is an old one, recently grown active through the development of adjoining lands proving the continuance of the rich Kearsarge lode under the Ahmeek tract. Is making from 20 to 25 tons of refined copper monthly from rock broken in sinking and drifting only, and will do no stoping until extensively opened, perhaps two years hence. The property is of exceptional promise and interesting because it shows a big Lake Superior copper-mine in an early stage of development.

Allouez.—The Allouez mine is at Allouez, Keweenaw county. James Chynoweth, superintendent. The company is an old one with a new mine. The old Allouez mine, opened on the Allouez conglomerate, was never profitable to the owners, and a second mine opened on the Osceola lode, did not promise well. Present development is by a shaft to tap the Kearsarge lode, which should cut that bed in October, 1904. As the Allouez owns the underlay, but not the outcrop of the Kearsarge, the shaft is being sunk to the lode at an angle of 80°, but on reaching it will be deflected to the angle of the amygdaloid. Back-guides will be used for the cages, which will have eight wheels each. Though not a pound of copper has been found yet in the new mine, it is universally conceded by local mining men to be a property of unusual promise.

Atlantic.—At Atlantic mine, Houghton county. Frank McM. Stanton, agent; Theo. Dengler, superintendent; F. G. Coggin, mill superintendent; A. D. Edwards, clerk; John Stratton, mining captain. Employs about 700 men. Has paid dividends of $940,000 and made 5,505,598 lb. of refined copper in 1903. Has a very complete mining-plant, including some powerful

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hoists, and is more frequently quoted than any other property in the United States or abroad as an example of an exceedingly low-grade mine, economically and successfully worked, the copper secured from the rock averaging only from 11 to 16 lb. per ton—0.55 to 0.80 per cent. of copper. The metal is distributed with exceptional uniformity throughout the rock, but is very fine and requires very great care in milling. The mill is at Redridge, on the shore of Lake Superior, and has six stamps. The Atlantic and Baltic make joint use of a steel gravity-dam across the Salmon Trout river, which impounds about 1,250,000,000 gal. of water. This dam, in five steel sections, anchored by its own weight in a concrete-base, is the first and, so far as known, the only structure of its kind.

Baltic.—At Baltic, 6 miles south of Houghton. Frank McM. Stanton, agent; F. W. Denton, superintendent; F. G. Coggin, mill superintendent; William C. Cole, clerk; Martin Trethewey, mining captain. The mine controlled by the Copper Range Consolidated Co. is one of the newer mines of the district, and has not yet obtained its full growth, though making 10,580,997 lb. of copper in 1903. It was the pioneer mine on the Baltic lode, the easternmost or oldest of the amygdaloid beds mined in the district. The lode is very wide, and an interesting and successful system of dry-walling with waste rock is employed underground. It has a good mining equipment, but not as heavy machinery as is found at the older and deeper mines. The stamp-mill is at Redridge, on Lake Superior, opposite that of the Atlantic.

Calumet & Hecla.—At Calumet, Houghton county. The mine was opened in a wilderness, and has seen a mining camp of 40,000 souls spring up about its shafts. James MacNaughton, superintendent; Will A. Childs and W. M. Gibson, assistant superintendents; W. H. Cake, mill superintendent; James B. Cooper, smelter superintendent; J. H. Lathrop, chief clerk; Fred S. Eaton, cashier; E. S. Grierson, chief engineer; Chas. Smith, smelter clerk; James Milligan, chief mining captain. The mine was opened in 1866 and has paid dividends to date of $85,850,000, paying at present quarterly dividends of $1,000,000 each. No other mining corporation in the world has paid an equal amount in profits. The largest payments were in 1899, when dividends were $10,000,000 during the

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calendar year. It produced 76,469,869 lb. of refined copper in 1903, being fourth among the world's producers in that year. A book, several times the size of the present paper would be required to give anything like a complete condensed description of this property, and an entertaining volume of several hundred pages might be written without exhausting the subject. Merely a few salient features can be touched upon in this brief description.

The Calumet & Hecla mine proper is opened on the Calumet conglomerate-bed for a distance of upwards of 2 miles along the outcrop. There are 12 shafts on the dip of the lode, 37° 30′, and one vertical shaft to the westward, this cutting the conglomerate at a depth of 3,287 ft., and being bottomed at a depth of 4,920 feet. The deepest incline shaft, No. 4 Calumet, has a depth of 8,100 ft. on the dip of the lode. The conglomerate-mine is divided into four sections, known as the Calumet, Hecla, South Hecla (or Black Hills) and Red Jacket. Outcropping at an average distance of 730 ft. to the eastward is the Osceola amygdaloid, on which are five shafts, Nos. 13 to 17, inclusive, while about one-half mile east of the conglomerate two new shafts, Nos. 19 and 20, are being sunk on the Kearsarge amygdaloid-bed. An interesting feature of mining-practice is a blind shaft, now being sunk to develop the "five forties," a tract of 200 acres lying between the Tamarack and Tamarack Junior mines, and inaccessible by any of the present shafts. This blind shaft starts from the drift at the bottom of No. 4 Calumet shaft, which has a vertical depth of 4,748 ft. and an actual depth of 8,100 ft. on the dip of the lode. To sink a new shaft below the bottom of an old shaft, itself more than a mile and a half in depth, is a novelty in mining-practice. The Calumet conglomerate varies from 8 to 40 ft. in width, averaging 12 ft. in cross-section, giving about 43,200 tons of stamp-rock per acre; or, deducting ground left in pillars, which are 75 ft. on either side of each shaft, there is enough rock under every average acre of surface to run the mill one week. The ground is treacherous and requires heavy timbering and constant watchfulness, taking about 30,000,000 ft. of timber annually for underground shoring. The mine has suffered serious losses from fire on five different occasions, and great vigilance is exercised to detect any incipient underground blaze. To fight mine-fires

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there are watchmen, pipe-lines and hydrants, with fire-hose, chemical extinguishers, fire-alarm systems, underground telephones and a system of fire-doors by which the mine can be divided into sections.

The machinery equipment of the mine is bewildering in its immensity. At No. 4 Calumet shaft is the 4,700-h.p. engine Superior and the 7,000-h.p. engine Mackinac, the latter operating a 500-drill air-compressor. There are also auxiliary engines of 2,000 and 600 h.p., and several smaller engines. The Hecla engine-house has engines of 600, 900 and 2,000 h.p., and a 144-drill air-compressor. The G. H. &. S. engine-house has two engines of 2,000 h.p. each, and the engine-house for Hecla shafts Nos. 7 and 8 has two 2,000-h.p. engines, operating 25-ft. drums by spur-gearing, and a 5,000-h.p. Leavitt engine, operating the man-cars. The Red Jacket shaft has an 8,000-h.p. quadruple hoist and a boiler-house with ten 1,000-h.p. boilers. There are many smaller engine-houses, all with powerful machinery. At the Red Jacket shaft the Kimberley system of swinging-skips under cages is in use, greatly increasing the hoisting-capacity.

The company owns about 1,200 dwelling-houses, and about 1,000 houses have been built by employees on the company's lands. Several complete fire-departments are maintained at various points. About 30 churches and a half dozen school-houses have been built on the company's land and are maintained for employees. The Calumet & Hecla Co. owns a fleet of steel steamers and a private ship-canal deep enough to float any vessel on the Great Lakes. Sawmills are owned and operated at various points, and a big auxiliary smelter is maintained at Buffalo, N.Y.

The stamp-mills are at Lake Linden, 4 miles from the mine, and connected therewith by a private railroad. The mills are in two sections, with 11 stamps each, the southernmost, or Calumet mill, having an addition of 6 stamps, giving a total of 28 stamps in both mills. The mills are being rebuilt of steel, in two sections each, about one year being required for reconstructing each section. The present milling-capacity of 6,000 tons per day will be increased to nearly 10,000 tons when the remodeling is completed, giving the Calumet & Hecla a crushing-capacity greater than the combined capacity of the Homestake, Treadwell and Simmer & Jack, the three gold-mines of

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the world having the largest mills. Interesting features at the mills are the pumps and sand-wheels. The pumping-engine, Michigan, of daily capacity 60,000,000 gal., is the largest in the world. The sand-wheels, used to elevate the sludge, are all giants, the largest being the new 64-ft. wheel in the Hecla wheelhouse. This wheel weighs 500 tons and is driven by a 700-h.p. dynamo. The smelting-plant is at Hubbell, a mile south of the mills at Lake Linden.

Centennial.—Just north of Calumet. James Chynoweth, superintendent; John Pentecost, mining captain. This mine is an old one remodeled. The original Centennial was opened on the northern extension of the Calumet conglomerate, which proved unprofitable. A second mine on the Osceola amygdaloid was but little better, and a third mine has been opened on the Kearsarge amygdaloid. The square mile owned by the Centennial company carried the underlay, but not the outcrop of the Kearsarge bed, hence a few acres carrying the outcrop were bought of another owner, and a 90-ft. right-of-way secured to connect the new tract, touching the old on the southeast corner. Two shafts were sunk, and in order to develop the entire square mile of underlay it was necessary to divert the northernmost shaft about 15° northward upon the plane of the lode after entering the main tract. Some delicate and novel engineering features were presented for solution in this work. The Centennial stamp-mill, at Grosse Pointe, Portage Lake, was built by the Arcadian company, which was the Standard Oil Company's first introduction to copper, and cost somebody about $3,000,000. The mill went into commission on Centennial rock in July, 1904.

Champion.—Dr. L. L. Hubbard, general manager; F. G. Coggin, mill superintendent; R. R. Seeber, engineer; John Broan, mining captain; M. J. Harrington, clerk. Mine is at Painesdale, 10 miles south of Houghton, and is a new property, first opened in 1899. Produced 10,564,147 lb. of refined copper in 1903 and paid dividends of $300,000 that year. The mine is opened on the Baltic lode, which is both wide and rich, and the property is of great merit and highest prospects. The general manager, Dr. Hubbard, who was formerly State Geologist, opened the mine he is now managing. Dr. Hubbard is an exception to the generally assumed rule that a thorough scien-

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tist does not make a practical mine-manager. Lake Superior can furnish exceptions to all bad rules.

Copper Range.—The Copper Range Consolidated Co. is not directly a producer of copper; being a securities-holding corporation owning the stock of the Baltic and Trimountain mines and a half interest in the Champion mine, also the Copper Range railroad, part of the Michigan smelter and extensive tracts of mineral-lands on the South Range. The construction of the Copper Range railroad (R. T. McKeever, general manager) has made possible the development of the great mineral territory lying between Portage Lake and the Ontonagon river, and the company has been the most important factor in the opening of the new mines of the Lake Superior copper district.

Franklin and Franklin Junior.—Both the Franklin and the Franklin Junior mines are owned by the Franklin Mining Co. The old Franklin is at Hancock, next north of the Quincy, and is merely a scram, its side-lines cutting off the lode, still payable at a depth of 3,200 feet. The Franklin Junior, 3 miles north, was originally the Albany & Boston, and later the Peninsula. The Junior is working a wide, hard, low-grade conglomerate, but is securing a large production, the property being managed with great prudence and zeal. The mill is at Grosse Pointe, adjoining that of the Centennial. Joshua D. Hosking, superintendent; Edward Warne, mill superintendent; Arno Jæhnig, clerk; Cyrus Truan and John Doney, mining captains.

Isle Royale.—At Houghton. W. E. Parnall, superintendent; R. M. Edwards, assistant superintendent; James G. Glanville, mill superintendent; H. D. Haddock, clerk. The Isle Royale affords a striking example of both the uncertainties and possibilities of Lake Superior copper-mining. The company was organized in 1897 and took over three old mines on the hill south of Houghton. Later, another new company with large landed area and capital was absorbed, giving the Isle Royale $2,000,000 in cash for development and equipment, after the lands were paid for. Of this amount $1,500,000 were used in opening a large new mine below the level of the old workings and building a modern mill. The equipment is of the best, the eastern management is experienced and successful, and the local management is equally good, but the mine proved a sore disappointment. Of the very small profit made last year, more

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than half came from interest on the company's $500,000 surplus. One of the two new shafts was burned December, 1903, and prospects for the mine were blue. The management, being experienced, did not haul down the flag, but began systematic exploration of its lands in search of something better. A diamond-drill gave a rich core from the Isle Royale lode about 2 miles south of the mine-workings, and a shaft, begun this season, has penetrated about 50 ft. of as rich amygdaloid-rock as ever has been opened in the district. The Isle Royale promises to make a very large and rich mine, despite its lack of success in the past.

Mass.—At Mass City, Ontonagon county. James M. Wilcox, superintendent; Charles H. Krause, mill superintendent; Thomas Hall, mining captain. This company is a consolidation of three early-day mines and two old prospects. The mill is at Keweenaw bay. The Mass has six parallel cupriferous beds, opened by adits and crosscuts at various levels. The company produced 2,576,447 lb. of refined copper in 1903.

Michigan.—At Rockland, Ontonagon county. Samuel Brady, superintendent; S. H. Brady, assistant superintendent; J. E. Vanse, engineer; J. C. Thomas, mining captain. Began production with one leased stamp at the Mass mill late in 1903. The mine is interesting by reason of presenting geological peculiarities found nowhere else in the district. The property includes the old Minnesota mine, opened on a contact-vein, which paid dividends of $1,820,000. A short distance north of the Minnesota vein is the Calico amygdaloid, on which three shafts have been sunk by the Michigan. Running from the Calico at surface, into the Minnesota at depth, is the North Minnesota fissure-vein, while another branch vein taking a similar course is found at depth. There are also foot-wall and hanging-wall veins. The old Minnesota mine was unwatered by drill-holes bored from each successive level of the Calico shafts, as it was sunk.

Michigan Smelter.—The Michigan smelting-works two miles west of Houghton, on Portage Lake, is a new plant, owned by the Baltic, Trimountain, Champion, Atlantic, Wolverine and other mines of the Stanton and Paine groups. It has an annual capacity of about 60,000,000 lb. of refined copper, and is the most modern metallurgical plant of the district, embody-

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ing a number of labor-saving devices and other improved features not employed heretofore in the district. The plant was designed by Frank Klepetko, the designer of the 6,000-ton Washoe plant at Anaconda, Mont., which is by far the largest in the world.

Mohawk.—At Kearsarge, Keweenaw county. The mine is the northernmost one opened on the Kearsarge lode and is under the same management as the Wolverine. Fred Smith, superintendent; Willard J. Smith, assistant superintendent; Barney S. Shearer, mill superintendent; F. William Hartman, engineer; Frank Getchell, clerk. A fine new mine, which produced 6,284,327 lb. of refined copper in 1903 and will become a dividend-payer soon, having accumulated a surplus exceeding $500,000. The mine is interesting because of a fissure-vein crossing the Kearsarge bed at right-angles, carrying mohawkite and keweenawite along the intersection of the stratified bed and fissure. The stamp-mill is at the mouth of Tobacco river, on Traverse bay, Lake Superior.

Osceola Consolidated.—Main mine office is at Osceola, just south of Calumet. William E. Parnall, superintendent; William J. Uren, assistant superintendent; A. Lincoln Burgan, mill superintendent. Property consists of four separate mines, of which the Tamarack Junior, opened on the underlay of the Calumet conglomerate, is idle. The Osceola mine proper is next south of the Calumet & Hecla, and was first opened on the conglomerate, only a few acres of which proved payable. The North and South Kearsarge mines, opened on the Kearsarge amygdaloid, are the principal producers and are properties of great merit. The Osceola made 16,059,636 lb. of refined copper in 1903, and will make about 21,000,000 lb. during 1904. The company has paid dividends of $4,441,900. The mills on Torch lake are very extensive, and the Osceola now ranks in tonnage stamped second only to the Calumet & Hecla. A 40,000,000-gal. pump, operated jointly for the Osceola and Tamarack mills, is an interesting feature.

Phœnix.—At Phœnix, Keweenaw county. William A. Dunn, superintendent; William L. Dunn, assistant superintendent; Dugald Stewart, clerk. Is the northernmost active copper-mine of the Lake district, and the only mine now working a fissure-vein. It has a one-stamp mill at the mine, being the

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only property now working that has its mill located elsewhere than on Lake Superior or one of its arms. It secures considerable heavy copper and a little stamp-rock.

Quincy.—At Hancock. Is known locally as "the Old Reliable." John L. Harris, superintendent; James W. Shields, mill superintendent; Thomas Whittle, mining captain; Charles K. Hitchcock, engineer; Will P. Smith, smelter superintendent; William Bath smelter clerk. Employs about 1,700 men. The mine was opened in 1848 and became a dividend-payer in 1862. It has paid dividends of $14,820,000 to date, ranking in profits disbursed second to the Calumet & Hecla. Made 18,498,288 lb. of refined copper in 1903. It has a magnificent surface-equipment, including an 8,000-h.p. hoist, extensive shops, railroad, etc. The mine is very deep, being but a trifle less than a mile on the dip of the lode, which is much sharper than at Calumet, to the northward. Some idea of the great depth of the mine may be gained by looking north from Houghton, at the hill crowned by No. 7 shaft-house, and reflecting that four-fifths of the depth of the mine is below the waters of Portage lake, while but one-fifth is above. No. 7 shaft is sunk on a catenary curve and is an excellent example of good engineering. The stamp-mills, at Mason, on Torch lake, are crushing about 3,400 tons of rock daily. Electric haulage is used extensively in the mine. The smelter is in Ripley, at the foot of the hill below the mine, on the Hancock side of Portage lake.

Tamarack.—At Calumet, William E. Parnall, superintendent; A. Lincoln Burgan, mill superintendent; John B. Watson, engineer; John T. Reeder, clerk. This mine is famous as the deepest one in the world. It has five shafts, all vertical, working the underlay of the Calumet conglomerate and part of underlay of the Osceola amygdaloid. It has paid dividends of $8,490,000 and produced 15,286,093 lb. of copper in 1903, but is now operating at a materially higher rate. No. 1 Tamarack shaft, when first planned to sink vertically for nearly one-half mile on the chance of cutting the conglomerate, was regarded by many as the dream of a madman, but it proved an entire success. No. 5 shaft, vertical, is the deepest in the world, being bottomed at 4,938 ft. It was started in August, 1895, and at that time the late Captain William E. Parnall, father of

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the present superintendent, stated that the shaft would be bottomed in the lode, at a depth of about 4,650 ft., about Jan. 1, 1902. The shaft actually cut the conglomerate on Dec. 20, 1901, at a depth of 4,662 ft., being but 12 ft. deeper and bottomed 11 days earlier than was predicted when the first sod was cut.

The machinery-equipment at all of the five vertical shafts is very heavy, being the most powerful at shafts Nos. 3 and 5. At No. 5 there is a 6,500-h.p. double hoist, and a duplicate hoist is being erected. There is also a 100-drill air-compressor at this point. No. 5 shaft is of interest because of its framing and other features, in addition to its stupendous depth. At No. 5, the combination shaft-house and rock-house, built of steel, on separate foundations yet forming one building, will repay study, and the hoisting-plant at No. 3 shaft is also worthy of inspection. The Tamarack has two stamp-mills on Torch lake, near those of the Osceola, equipped with modern crushing- and washing-machinery.

Tamarack-Osceola Copper Manufacturing Co.—At Dollar bay, 4 miles east of Hancock. The plant includes a smelter, handling the mineral of the Osceola, Tamarack and Isle Royale mines, and a manufacturing-plant, making sheets and wires, the latter being the only copper-rolling and drawing-mill in the northwest.

Trimountain.—At Trimountain, 8 miles south of Houghton and midway between the Baltic and Champion mines. F. W. Denton, superintendent; John Knox, assistant superintendent; Edward Kœpel, mill superintendent; H. T. Mercer, engineer; Benjamin D. Noetzel, clerk. This mine, a new one, opened on the great Baltic lode, produced 9,237,051 lb. of refined copper in 1903, but is now being operated at a higher rate. It paid dividends of $300,000 in 1903 previous to its absorption by the Copper Range Consolidated Co. The mine presents many of the same features to be found at the Baltic and Champion mines, with which it is closely affiliated, and has the same wide and rich copper-bearing amygdaloid. The surface-plant is extensive and the property is managed with much vigor and success. The stamp-mill is at Beacon hill, 2 miles west of Redridge, on Lake Superior, with 4 stamps and a 20,000,000-gal. pump taking fresh water from a well connected with Lake Superior by a 40-in. riveted-steel pipe running to a crib in the lake 1,400 ft. from the shore.

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Victoria.—At Victoria, Ontonagon county. This mine is the westernmost of the active mines of the Lake Superior copper-district. Thomas Hooper, superintendent; George Hooper, mining captain. Extensive underground development has been secured, and the property is now awaiting a mill. The large natural water-power at Glenn Falls, on the west branch of the Ontonagon river, a short distance from the mine, is being improved, and several thousand horsepower will be delivered by means of a canal to divert the water from the river to a 20 by 20 ft. shaft 300 ft. in depth. Instead of using pen-stocks and turbines, the water will be made to compress air at the bottom of the shaft, affording a novel hydraulic air-compression plant. The topography at the Victoria is much bolder than in Houghton county, and the scenery about the Ontonagon county mines is unusually striking.

Winona.—At Winona, Houghton county, 25 miles southwest of Houghton. Frank McM. Stanton, superintendent; J. O. Peterson, mining captain; William Van Orden, clerk. This mine produced 1,036,944 lb. of refined copper in 1903, though still in the development stage. It has a fair plant of mining-machinery, suited to the present requirements, and sends rock to the Atlantic mill for reduction.

Wolverine.—At Kearsarge, Houghton county. Fred Smith, superintendent; Willard J. Smith, assistant superintendent; B. S. Shearer, mill superintendent; F. William Hartmann, engineer; William Pollard, mining captain, This small, but very rich, property is most excellently handled, securing from 29 to 30 lb. of refined copper per ton of rock stamped, which is the largest average return from any amygdaloid mine of the district, and second only to the Calumet & Hecla. It produced 9,024,034 lb. of refined copper in 1903 and has paid dividends of $1,770,000 to date. It's stock sold at $2.25 per share 10 years ago, and the mine now pays $8 a year in dividends. The stamp-mill at the mouth of the Tobacco river, on Traverse bay, Lake Superior, is a duplicate of the mill of the Mohawk mine.

Ishpeming, Mich.

The two special trains bearing the members and guests of the Institute left Houghton early on Tuesday evening, and ar-

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rived at Ishpeming about midnight, where the cars were placed on a siding and a much-needed quiet night's rest was obtained. The night was clear, cool and delightful, fully typifying the glorious fall climate of the Northwest. Early on Wednesday, September 20th, the Ishpeming Local Committee took the party in hand, and began the tour of inspection by visiting the great Lake Angeline basin, which was once the bottom of a lake, but is now the roof of three important mines operated by different companies. There is a thick coating of mud above the rock-roof of the mines and below the bed of the lake which is so soft that it will run like water if given an opportunity to make its way into the workings under the lake; and the mining-methods have been so carefully and successfully worked out with reference to this unusual factor, that but little trouble has resulted therefrom. Returning from the Lake Angeline basin, the party took a special train on the Lake Superior & Ishpeming railroad and proceeded to Marquette to inspect the charcoal furnace and chemical plant of the Pioneer Iron Co. at Marquette.

The following description of these works has been condensed from the pamphlet prepared by the company for its exhibit of the models at the St. Louis Louisiana Purchase Exposition, St. Louis, Mo. (Copies of this interesting pamphlet, together with one prepared by the Ishpeming Local Committee, containing the condensed itinerary and a short illustrated sketch of the points of historic interest on the Marquette iron-ore range were given to each member and guest of the Institute.)

The boiler-plant, consisting of 12 water-tube boilers of the Stirling type, aggregating 3,600 h.p., is so arranged that the furnace-gas may be consumed under a portion of them, according to its volume and fuel-value. There are times in the working of every charcoal-furnace when these gases are more than sufficient to supply the needs of the furnace proper, and when such is the case the chemical plant gets the benefit of the excess. The non-condensible gases coming from the kilns through the gas-main are also handled under these boilers. Tar, which has been separated by gravity from the pyroligneous acid, is led to the boilers through brass pipes to be used at intervals when the gases of either kind are least efficient; and its high value as a fuel makes itself felt immediately on its application. Eight of these boilers are equipped with Murphy automatic

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stokers, which, with their train of receiving-hopper, coal-crusher, elevators, conveyers and overhead magazine, reduce the labor of firing to a minimum.

The ash drops into hoppers, which spill into dump-cars, and is finally elevated into a storage-bin, whence it is loaded into side-dump cars to be used for filling where needed.

The charcoal breeze, of which there is a considerable amount, is mixed with the crushed bituminous coal, and plays no insignificant part in the general economy.

Another, and a very important factor in this economic scheme, is the extraction of the heat passing off from the boilers—which ordinarily goes to the smoke-stack at a temperature of about 500° F.—by the medium of Green economizers, and imparting it to the boiler-feed water, raising it to a temperature of 270° F., as compared with 180° F., usually obtained by other methods. In connection with the Green economizers is the Sturtevant induced-draft system, which enables the operator to regulate the draft with nicety at all times.

The plant, in full operation, yields daily about 130 tons of charcoal-iron of various grades, about 140 tons of charcoal, 20,000 lb. of acetate of lime, 2,500 gal. of wood-tar and 1,200 gal. of refined water-white methyl alcohol of 95 per-cent. strength.

The furnace, 70 ft. high and 12 ft. in diameter at the bosh, is equipped with bosh-jackets of the Farrell type. The blast is heated in three Cowper-Roberts regenerative stoves to a uniform temperature of about 1.250° F.

The engine-room is equipped with a disconnected cross-compound Southwark blowing-engine, having a high-pressure cylinder 22 by 36 in., a low-pressure cylinder 46 by 36 in. and two air-cylinders 48 by 48 in. in diameter.

In the same room with the blowing-engine are two direct-connected compound 150-kw. Westinghouse electric generators. The pump-room, which communicates with the engine-room through open arches, is at a lower level, and has three Prescott triple-expansion pumps, each of a capacity of 5,000,000 gal. in 24 hours. Two of these pumps are continually used to supply the furnace and chemical works with water for cooling-purposes, the third being held in reserve. The daily requirement of the furnace is 2,000,000 gal., and of the chemical

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works, 8,000,000 gallons. A Weiss condenser of a capacity of 21,000 lb. of steam per hour forms a central condensing-plant for all the steam-engines of the works. The stock-house, 100 by 30 ft., is constructed of wood and steel, and can be completely closed on all sides in periods of extreme weather—a feature of great importance in the severe northern climate. Two trestles traverse the entire length of the building, one arranged to dump ores on to a cement floor, which is circumscribed by a track carrying the charging-car and electric motor; the other carries center-dumping charging-cars over a series of steel bins in the side of the stock-house near the furnace. A simple device, designed by Mr. Farrell, is used during extremely cold weather to overcome the freezing of the ores in the stock-bins. The electric charging-car is equipped with a scale on which the ore and limestone are weighed as they are dumped into the car from the bins. The filled car is run to the foot of the furnace-incline, where the charge is tipped sideways into the self-dumping skip which carries it to the top of the furnace. The dry, hard-wood charcoal which forms the fuel of the furnace is shoveled into 20-bushel buggies, wheeled direct into the kiln, and the filled buggies then wheeled to flat-cars, which are brought by the electric motor to the incline, where the charcoal is tipped into the skip.

The furnace plant and chemical works are connected with a system of tracks aggregating about 3.5 miles, which is so complete that haulage by horse is used solely to remove the slag and coke-breeze.

The chemical plant contains a battery of 86 kilns, each holding 80 cords of wood, arranged along two trestles on which are received the cars loaded with cord-wood as they come from the forest.

The kiln-system for carbonizing wood was adopted by the company after an exhaustive study of all other methods, which showed that, in the saw-mill refuse or other very cheap fuel, any form of retort furnace could not be operated with financial success.

The daily requirement is 20 carloads of 16 cords each, which are first unloaded into the kilns and the wood then snugly arranged in ranks by 20 men.

After the kiln is packed full of wood, fire is introduced in

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the circular opening at the apex of its dome, and as soon as it gains sufficient headway this opening is closed and the smoke generated escapes through a flue leading from the base of the kiln and communicating with its chimney. This process continues until the aqueous vapors have largely escaped into the atmosphere, and the gases from the kiln are of sufficient value to warrant their condensation.

At this stage, the top of the chimney is closed and a damper are opened which completes the communication from kiln to condensers through the medium of the smoke-main.

At the condenser-end of the smoke-main, electrically-driven fans induce a gentle draft on the kilns, and also subject the gases to a slight pressure while passing through the condensers, and it is this contact of the gases with the cold surface of the copper tubes of the condenser which precipitates the condensible portions of the gases—the non-condensible portion going to the boilers to be burned as fuel as already described.

It will be a revelation perhaps to the uninitiated to learn that the liquor product of the condensible gases from one cord of wood amounts to about 224 gal., or more than the contents of four large kerosene-barrels. This liquor, which is known in the industry as "green liquor," or pyroligneous acid, is largely made up of water, but carries alcohol, tar, ammonia compounds, acetone (amyl-alcohol, ethers, aldehydes), and acetic acid, the alcohol, acetic acid (as acetate of lime) and formaldehyde being the only commercial products secured at this plant.

The heavier elements of tar, which would be very objectionable in subsequent processes, are separated by gravity in liberal settling-tanks placed at a suitable elevation whence it is sent to a higher elevation to be injected by steam as a spray under the boilers and used as fuel.

The liquor which remains in the settling-tanks is of large volume, and the elevating of this material involves the expenditure of considerable power. For this reason, the designers of other plants have sought to raise the liquor but once in the process, and thereafter to maintain desired elevations by the rising of vapors, or by gravity; but this idea necessitates the placing of the first system of stills—the primaries—at some height, and a consequent increase in the height and cost of the building which contains them. In the case before us, the en-

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gineers have made use of a source of power which had before been overlooked, namely, water-power obtained by installing turbines, and using the overflow-water from the condensers. Having this power, the designers were able to carry out certain economic schemes which would not have been warranted without it. It is usual to assemble stills of all kinds in one building, but, in this case, the distilling of the pyroligneous acid and the neutralizing of the distillate with milk of lime is conducted in the primary still-house, designed especially for these processes.

By reason of the acid character of the material to be distilled, these stills are made of heavy copper, and make up a considerable percentage of the cost of a plant of this kind. The function of primary stills is to throw over to be condensed, vapors of alcohol and acetic acid, both of which are more volatile than water (though a very considerable amount of water-vapor passes over with them), and leaving behind, as a residuum, water containing tarry products and coloring matter; the condensed product of the still being correspondingly lessened in volume and raised in strength of alcohol and acid.

The neutralizing tanks receive this product, which is vigorously agitated by vertical shafts bearing wings, and connected with water-power, while the milk of lime is being introduced. The result is an intimate chemical union between the acid and as much of the lime as it is capable of carrying, and finally a perfectly neutralized mixture.

This neutralized liquor is now allowed to settle, all excess of lime and some foreign material depositing itself on the floor of the tank as core, and the brilliant claret-colored portion is sent to the refining-still house for additional treatment, which eliminates further water and the lighter oils that are very persistent, carrying coloring matter and objectionable odors, and finally to produce a 95-per-cent. methyl alcohol free from objectionable odors, and permanently water-white in color.

The stills which are used to produce this result are termed intermediate or lime-lee stills, and refining stills. The accessories of these stills are many and varied. A condenser in some form is common to all of them, but the appliances for eliminating the water, and saving the valuable alcohol, may take the form of the goose, the pans, or series of pans arranged in a great variety of combinations in columns of various designs. What-

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ever form these appliances may take, the governing principle is that of fractional distillation.

By the operation of the intermediate or lime-lee stills, the acetic acid combined with lime was left behind to be treated for acetate of lime; and the distillate known as crude alcohol remains to be refined. Both in the intermediate and refining processes, advantage is taken of the physical law that alcohol is more volatile than water, and that if the goose, the pans, or the columns are kept at a certain temperature, the alcoholic vapors will remain as vapors, and pass over to the condenser, while the aqueous vapors will be condensed and returned to the still as water. This is what is meant by fractional distillation—a principle which is employed wherever the recovery of alcohols, whether of the methyl or ethyl group, is the object.

A log of hard maple, representing in quantity just one-sixteenth of a cord of wood, is equivalent in charcoal obtained by this process to 55 lb.—the quantities of by-products obtained from this amount of wood are, 13 gal. of pyroligneous acid, 0.5 gal. of wood-tar, 4 lb. of gray acetate of lime, and 0.25 gal. of 95 per-cent. refined wood-alcohol.

Memoranda relating to the operations of the Cleveland-Cliffs Iron Co., Pioneer Iron Co., and their allied interests:

Iron-ore mined in 1903 (gross tons),	1,649,567
Total mined up to 1903 (about),	18,000,000
Charcoal pig-iron made in 1903,	81,444
Charcoal pig-iron made 1854–1903 (about),	721,000
Wood-alcohol made in 1903 (gallons),	446,423
Acetate of lime made in 1903 (net tons),	3,000
Miles of standard gauge railroad operated,	228
Tons carried in 1903 on railroad operated,	2,212,219
Ships owned, steam and sail,	8
Gross tons freight carried,	481,392
Average men employed 1903,	3,500
Average wages at mines, per day,	$2.21
Average wages on vessels,	$2.23
Average wages at furnaces,	$2.20
Acres of land owned and controlled,	1,400,000

All of the iron-ores and the wood used by the Pioneer Iron Co. are furnished from the mines and lands of the Cleveland-Cliffs Iron Co. and over railroads owned or controlled by the same interests.

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The officers of the Cleveland-Cliffs and the Pioneer Furnace Co. are,—Wm. G. Mather, President and Treasurer; J. H. Wade, Vice-President; J. H. Sheadle, Secretary; R. C. Mann, Auditor; M. M. Duncan, Agent Dept. Mines and Minerals; Austin Farrell, Manager Furnace Dept.; Samuel Redfern, Agent Land Dept.; H. R. Harris, General Manager Lake Superior & Ishpeming Ry. Co., Munising Ry. Co., Marquette & Southeastern Ry., Marquette, Mich.

Early in the afternoon, directly after the inspection of the furnace plant and chemical works, a luncheon was tendered to the party at the Wawonowin Golf club house, which was tastefully decorated with autumn leaves of brilliant and variegated hues. A cordial address of welcome was given by the Hon. H. O. Young, Member of Congress from the Upper Peninsula, to which Dr. Raymond fittingly responded in his usual delightful manner. At the close of the luncheon, carriages and horses were provided to take the members and guests to various places where they wished to go, and the remainder of the day was passed in drives about Ishpeming and environs, to Marquette and along the beautiful cliff road. Some of the members accepted the courteous invitation of the Cleveland Cliffs Co., and went underground in the Cliffs Shaft mine.

In the evening, prior to the starting of the trains, an interesting reception was held in the club car, which though informal was all the more hearty and enjoyable. The aforementioned band of Ojibway Indians entertained our hosts with an impromptu dance and appropriate songs to the sad disarrangement of their vocal chords on the morrow. Finally the trains started, to the enlivening music of the band which tendered the party "God Speed" in music.

The run to Chicago was made on time and the arrival of the trains at the Union Depot, Thursday morning, September 22, completed the trip to the Lake Superior iron- and copper-districts. A number of the party went direct from Chicago to St. Louis, to attend the supplementary session of the Institute, held there Saturday, September 24, 1904, while others spent the intervening time in Chicago and vicinity.

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St. Louis, Mo.

At St. Louis the headquarters of the Institute were at Block No. 74, in the Mines and Metallurgy Building. On Saturday morning, September 24th, a number of the members and guests availed themselves of the opportunity to attend the meeting in the Mining Engineering Section of the International Congress of Arts and Science at Hall No. 11, of the Hall of Congresses, adjoining the Administration Building. Mr. John Hays Hammond was Chairman, and Dr. Joseph Struthers, Secretary of this Section. Addresses were made by Prof. Robert H. Richards, of Boston, Mass.; Prof. Samuel B. Christy, of Berkeley, Cal.; James D. Hague and Dr. R. W. Raymond, of New York, N. Y.; notice of which has already been recorded in the early part of these proceedings.

In the evening an informal reception and smoker was given in the Missouri State Building to the members of the Institute by the St. Louis Engineers' Club. A souvenir book, prepared by the Engineers' Club of St. Louis and vicinity, was distributed to the members of the party at the headquarters of the Institute.

By special arrangement in honor of the Institute, the Mines and Metallurgy Building was opened for private inspection on Sunday morning, September 25, for the benefit of the members and guests present, and the quiet view of the exhibits, unhampered by the usual crowd of visitors, was fully appreciated and enjoyed, as was also the informal reception given by the Kentucky State commission at its exhibit. A luncheon was served in the Assembly Room, preceded by a short address of hearty welcome by David R. Francis, President of the Governing Board of the Exposition, and a few remarks by Director F. J. v. Skiff.

Directly after disposing of the good things that had been spread before the party for both mental and bodily refreshment, a trip through the mining-gulch was made and the various exhibits reviewed. Later, automobiles conveyed the party around the Exposition grounds and along the principal boulevard and Forest City Park drives. This pleasant feature closed the official visit of the Institute, but many of the members remained at the Exposition for several days, and profited still further by the kind and assiduous attention of the various

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members of the local committee, who, although they have already received the official thanks of the Council of the Institute, are hereby given a public acknowedgment of the many courtesies and favors extended by them to individual members and guests.

Members and Guests Registered.

The following is a list of the members and guests who visited the iron-ore regions of Minnesota. In addition to these names, there were a large number of the citizens of Duluth and vicinity present at other portions of this trip, whose cordial hospitality did much to make the social features of the visit to Duluth so successful, and whose names were unfortunately not registered at the headquarters of the Institute.

G. E. Abbott,	Eveleth, Minn.
J. B. Adams,	Duluth, Minn.
W. C. Agnew,	Duluth, Minn.
Ward Ames, Sr.,	Duluth, Minn.
H. F. Bain,	Washington, D. C.
W. A. Barrows, Jr.,	Sharpsville, Pa.
George Bartol,	Cleveland, Ohio.
Herr Baum,	Essen, Germany.
Theodore Beckert,	Breslau, Germany.
Master A. R. Bellenger,	Syracuse, N. Y.
J. A. Capp,	Schenectady, N. Y.
J. M. Clements,	New York, N. Y.
T. F. Cole,	Duluth, Minn.
G. L. Collord,	Clairton, Pa.
C. A. Congdon,	Duluth, Minn.
Joseph B. Cotton,	Duluth, Minn.
H. W. Craver,	Pittsburg, Pa.
Mrs. H. W. Craver,	Pittsburg, Pa.
D. T. Croxton,	Cleveland, Ohio.
S. W. Croxton,	Cleveland, Ohio.
H. L. Dresser,	Duluth, Minn.
Theodore Dwight,	New York, N. Y.
Prof. Howard Eckfeldt,	South Bethlehem, Pa.
Augustus H. Eustis,	Boston, Mass.
F. A. Eustis,	Boston, Mass.
T. M. Eynon,	Philadelphia, Pa.
B. F. Fackenthal, Jr.,	Easton, Pa.
Mrs. B. F. Fackenthal, Jr.,	Easton, Pa.
H. C. Farnum,	Detroit, Mich.
Paul R. Forbes,	New York, N. Y.
W. Frentzel,	Essen, Germany.
James Gayley,	New York, N. Y.
F. W. Getty,	Chicago, Ill.
Ralph Gillette,	Minneapolis, Minn.

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Rudolf F. Gœtze,	Bochum, Germany.
R. R. Goodrich,	Westfield, N. J.
Jules Goujon,	Moscow, Russia.
J. B. Guinn,	Joplin, Mo.
G. G. Hartley,	Duluth, Minn.
C. S. Hill,	New York, N. Y.
Levi Holbrook,	New York, N. Y.
Mrs. Levi Holbrook,	New York, N. Y.
F. E. House,	Duluth, Minn.
Nelson P. Hulst,	Duluth, Minn.
James C. Hunter,	Duluth, Minn.
E. S. Hutchinson,	Newtown, Pa.
F. G. Jewett,	Hibbing, Minn.
Julian Kennedy,	Pittsburg, Pa.
Mrs. Julian Kennedy,	Pittsburg, Pa.
C. W. Kieswetter,	Duluth, Minn.
Paul S. King,	New York, N. Y.
H. W. Klein,	Dahlbruch, Germany.
Fried Kœnig,	Eisleben, Germany.
George J. Krebs,	Somerset, Pa.
Dr. H. B. Kümmel,	Trenton, N. J.
Mrs. H. B. Kümmel,	Trenton, N. J.
W. G. LaRue,	Baraboo, Wis.
Mrs. W. G. LaRue,	Duluth, Minn.
Prof. C. K. Leith,	Madison, Wis.
Prof. J. Lodge,	Tuscaloosa, Ala.
Mrs. J. Lodge,	Tuscaloosa, Ala.
E. J. Longyear,	Minneapolis, Minn.
Max Ludwig,	Eisleben, Germany.
Jawood Lukens,	Conshohocken, Pa.
Mrs. Jawood Lukens,	Conshohocken, Pa.
J. F. Lutes,	Biwabik, Minn.
W. A. McGonagle,	Duluth, Minn.
H. H. McNamara,	Duluth, Minn.
D. B. Meacham,	Cincinnati, Ohio.
Hans Mentzel,	Bochum, Germany.
W. N. Merriam,	Duluth, Minn.
L. J. Merritt,	Duluth, Minn.
George Mesta,	Pittsburg, Pa.
Miss A. A. Mesta,	Pittsburg, Pa.
Miss Mills,	New York, N. Y.
C. H. Munger,	Duluth, Minn.
C. M. Myrick,	San Francisco, Cal.
J. F. Myrick,	San Francisco, Cal.
R. V. Norris,	Wilkes-Barre, Pa.
W. J. Olcott,	Duluth, Minn.
Miss M. Ormrod,	Allentown, Pa.
Professor Osann,	Clausthal, Germany.
Thomas Owens,	Duluth, Minn.
George W. Pfeiffer,	Daiquiri, Cuba.
S. H. Pitkin,	Akron, Ohio.
L. W. Powell,	Duluth, Minn.

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Dr. R. W. Raymond,	Brooklyn, N. Y.
Mrs. R. W. Raymond,	Brooklyn, N. Y.
Miss Raymond,	Brooklyn, N. Y.
E. F. Remer,	Stevenson, Minn.
Mrs. G. B. Richards,	Buffalo, N. Y.
Master T. D. Richards,	Buffalo, N. Y.
Dr. H. Ries,	Ithaca, N. Y.
A. J. Rossi,	New York, N. Y.
Mr. Salsich,	Duluth, Minn.
F. Z. Schellenberg,	Pittsburg, Pa.
E. J. Schmitz,	New York, N. Y.
John Uno Sebenius,	Duluth, Minn.
Amos Shepherd,	Hibbing, Minn.
J. M. Sherrerd,	Easton, Pa.
A. T. Shurick,	Las Esperanzas, Mexico.
John Stokes,	Washington, D. C.
Dr. Joseph Struthers,	New York, N. Y.
H. B. Sturtevant,	Duluth, Minn.
Mrs. H. B. Sturtevant,	Duluth, Minn.
George D. Swift,	Duluth, Minn.
Mrs. George D. Swift,	Duluth, Minn.
Harry Thomas,	Hibbing, Minn.
Kirby Thomas,	Superior, Wis.
Mrs. K. Thomas,	Superior, Wis.
R. R. Trezona,	Eveleth, Minn.
B. J. Vallatt,	Virginia, Minn.
A. H. Viele,	Duluth, Minn.
Mrs. A. H. Viele,	Duluth, Minn.
G. A. Voskühle,	Wynberg, South Africa.
Olof Wenstrom,	Boston, Mass.
Mrs. C. Whitaker,	Allentown, Pa.
De B. Whitaker,	Daiquiri, Cuba.
William Wilkins,	Ashland, Wis.
F. S. Witherbee,	Port Henry, N. Y.
H. J. Wolff,	Telluride, Colo.
Howard Wood,	Philadelphia, Pa.
Miss M. B. Wood,	Philadelphia, Pa.
Miss Rachel B. Wood,	Philadelphia, Pa.
Walter Wood,	Philadelphia, Pa.

The following list, which in all probability does not contain the names of all who attended the session and excursions, comprises the names registered at the headquarters of the Institute at the St. Louis Exposition.

Jiro Abratani,	New York, N. Y.
H. H. Albert,	Berlin, Germany.
H. F. Bain,	Washington, D. C.
A. K. Baker,	Leadville, Colo.
Karl Baryhoff,	Berlin, Germany.
C. E. W. Bateson,	New York, N. Y.

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Robert Bell,	Ottawa, Ontario, Can.
Theodore Beckert,	Breslau, Germany.
O. M. Bilharz,	Flat River, Mo.
Arthur K. Bohn,	Leadville, Colo.
Henry Brooke,	Thayer, W. Va.
Lucius P. Brown,	Nashville, Tenn.
M. R. Campbell,	Washington, D. C.
J. Morgan Clements,	New York, N. Y.
F. L. Clerc,	Denver, Colo.
Samuel B. Christy,	Berkley, Cal.
Pierre Chouteau,	St. Louis, Mo.
F. T. Connor,	Burlington, Iowa.
A. R. Crook,	Evanston, Ill.
Floyd Davis,	Des Moines, Iowa.
G. J. Dawbarn,	Ballarat, Australia.
H. T. Dickinson,	Kimberley, South Africa.
Earl Dissinger,	Deadwood, So. Dak.
E. V. d'Invilliers,	Philadelphia, Pa.
R. M. Draper,	Great Falls, Mont.
J. A. Ede,	La Salle, Ill.
E. E. Ellis,	Evanston, Ill.
Augustus H. Eustis,	Boston, Mass.
F. A. Eustis,	Boston, Mass.
B. F. Fackenthal, Jr.,	Easton, Pa.
Mrs. B. F. Fackenthal, Jr.,	Easton, Pa.
Oscar Falkman,	Stockholm, Sweden.
H. E. Farnum,	Detroit, Mich.
Geo. D. Fisher,	Chicago, Ill.
H. E. Flewellin,	New York, N. Y.
E. L. Foucar,	San Francisco, Cal.
F. Julius Fohs,	Marion, Ky.
F. Pauline Fohs,	Marion, Ky.
Frederic B. Fohs,	Marion, Ky.
John H. Frick,	Warrensburg, Mo.
John A. Garcia,	St. Louis, Mo.
Frank Glass,	Butte, Mont.
Rudolf F. Gœtze,	Bochum, Germany.
R. R. Goodrich,	Westfield, N. J.
C. H. Gorill,	Oakland, Cal.
J. S. Grasty,	Baltimore, Md.
Karl Grossmann,	Kulebaki, Russia.
John Hays Hammond,	New York, N. Y.
James D. Hague,	New York, N. Y.
Jas. O. Handy,	Pittsburg, Pa.
E. Harms,	Rorreon, Mexico.
C. W. Hayes,	Washington, D. C.
V. C. Heikes,	Denver, Colo.
Henry D. Hibbard,	Plainfield, N. J.
John E. Hodge,	Baraboo, Wis.
J. A. Holmes,	St. Louis, Mo.
Oscar Holversen,	Northfield, Minn.
Levi Holbrook,	New York, N. Y.

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Mrs. Levi Holbrook,	New York, N. Y.
Richard L. Humphrey,	Philadelphia, Pa.
Edward S. Hutchinson,	Newtown, Pa.
Joseph Hutchinson,	Pottstown, Pa.
Clyde H. Jay,	Colorado Springs, Colo.
R. I. Johnson,	Minneapolis, Minn.
A. R. Justice,	Philadelphia, Pa.
G. E. Keckzie,	Durango, Mexico.
W. Oakes Kibble,	Denver, Colo.
Paul F. King,	New York, N. Y.
Geo. Kislingbury,	Los Angeles, Cal.
A. Kowoirth,	St. Louis, Mo.
George J. Krebs,	Somerset, Pa.
W. H. Landers,	Galena, Ill.
Mark R. Lamb,	San Diego, Cal.
George M. Lehman,	Pittsburg, Pa.
J. H. Lee,	Baltimore, Md.
E. C. Limbach,	Gilt Edge, Mont.
Max Ludwig,	Eisleben, Germany.
Jawood Lukens,	Conshohocken, Pa.
Mrs. Jawood Lukens,	Conshohocken, Pa.
F. W. Lyman,	Minneapolis, Minn.
Lesley McCreath,	Harrisburg, Pa.
Carlo Massa,	Genova, Italy.
Lawrence May,	St. Louis, Mo.
George Meryweather,	Chicago, Ill.
Hans Mentzel,	Boehm, Germany.
Lewis Minon,	Birmingham, Ala.
J. T. Monell,	Flat River, Mo.
Phillip N. Moore,	St. Louis, Mo.
Frederic H. Morley,	Denver, Colo.
H. Alfred Moeckel,	Zwickaw, Saxony.
C. A. Murdock,	Butte, Mont.
C. M. Myrick,	San Francisco, Cal.
Prof. A. Vander Naillen,	San Francisco, Cal.
R. V. Norris,	Wilkes-Barre, Pa.
B. C. Oliphant,	Oil City, Pa.
M. Otagawa,	Tokio, Japan.
A. C. Overpeck,	Hill City, So. Dak.
E. W. Parker,	Washington, D. C.
F. W. Parsons,	Rush Run, W. Va.
C. E. Peet,	Chicago, Ill.
George W. Pfeiffer,	Daiquiri, Cuba.
B. R. Putnam,	Butte, Mont.
R. M. Raymond,	El Oro, Mexico.
Mrs. R. M. Raymond,	El Oro, Mexico.
Dr. R. W. Raymond,	New York, N. Y.
Mrs. R. W. Raymond,	New York, N. Y.
Miss Raymond,	New York, N. Y.
T. T. Read,	New York, N. Y.
Robert H. Richards,	Boston, Mass.
Georges Riviere,	Quimperle, France.

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S. W. Russell,	Deadwood, So. Dak.
J. J. Rutledge,	Baltimore, Md.
Haven Sawyer,	Gazelle, Colo.
F. Schmite,	Kalk by Cöln, Germany.
T. E. Schwarz,	Denver, Colo.
Gust. Sessinghaus,	Denver, Colo.
J. M. Sherrerd,	Easton, Pa.
Sheridan Smillie,	New York, N. Y.
George Steiger,	Washington, D. C.
Dr. Joseph Struthers,	New York, N. Y.
Theodore Sutro,	New York, N. Y.
K. Takenouchi,	Kosaki, Japan.
Robert Teske,	Berlin, Germany.
Arthur Thacher,	St. Louis, Mo.
W. H. Thomas,	Elyria, Ohio.
Priestly Toulmin,	Lehigh, Ala.
E. O. Urich,	Washington, D. C.
F. B. VanHorn,	Rolla, Mo.
F. G. Waffler,	Berlin, Germany.
Y. Watanabe,	Tokio, Japan.
A. E. Wheeler,	Great Falls, Mont.
H. A. Wheeler,	St. Louis, Mo.
Olof Wenstrom,	Boston, Mass.
S. Weidman,	Madison, Wis.
William H. Wiley,	East Orange, N. J.
F. S. Witherbee,	Port Henry, N. Y.
Harry J. Wolf,	Telluride, Colo.
Howard Wood,	Philadelphia, Pa.
Miss M. B. Wood,	Philadelphia, Pa.
Miss Rachel B. Wood,	Philadelphia, Pa.
Walter Wood,	Philadelphia, Pa.
L. H. Wygant,	Denver, Colo.
Bruce C. Yates,	Lead, So. Dak.
J. E. Zellweger,	St. Louis, Mo.

NOTE BY THE SECRETARY.—Comments or criticisms upon all papers, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

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Biographical Notices of 1903.*

THE following paragraphs, constituting the concluding portion of the Annual Report of the Council for 1903, have been withheld from publication until now, in order to make them as accurate and complete as possible. They will be published in Volume XXXV. of the *Transactions* in their proper place at the end of the said Annual Report.

Franklin Ballou was born in Pittsburg, Pa., May 29, 1845. He was the son of an old resident, Arthur Ballou, a manufacturer and wholesale dealer in vinegar. In early youth Franklin attended the Western University of Pittsburg, but left school at the age of 16 to become book-keeper for his father, upon whose retirement from business, in 1868, he associated himself with Walter C. Childs, of Pittsburg, in the manufacture of furnace-grates and asbestos-cement for blast-furnaces. This firm was known as W. C. Childs & Co., and did a successful business for several years. In 1869 Mr. Ballou married Miss May S. Wilson, daughter of Judge Wilson, of Lewistown, Pa.

In 1876 he spent several months traveling in the West and studying the mining industry, which greatly interested him. In consequence of this journey, he wound up his business in Pittsburg and removed in 1877 to Georgetown, Colo., where for several years he conducted a sampling-works, and built up an extensive business as an agent for the sale of ores, besides engaging to some extent in mining enterprises. It is said that he controlled at one time about one-eighth of the output of the Leadville mines. In the early 80's, he became Vice-President and General Manager of the Bimetallic Smelter at Leadville, of which Mr. David H. Moffat was the principal owner. This was practically the first successful demonstration of pyritic smelting in Colorado. Upon the sale of the works in 1897 to the American Smelting and Refining Co., Mr. Ballou retired

* See p. 14 of the pamphlet edition of the Proceedings of the Eighty-sixth (Thirty-fourth Annual) Meeting, Atlantic City, N. J., February, 1904.

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from active business, with a comfortable fortune and a recognized position among the leading metallurgical managers of Colorado. Moreover, his social sympathy, generosity and unswerving integrity had won for him the cordial esteem and confidence of his associates and acquaintances.

Mr. Ballou was naturally a student, and, outside of metallurgy, was deeply interested in scientific, philosophical and theological subjects. At the age of 25 he encountered the theological writings of Swedenborg, which so appealed to him that he connected himself with the New Church, and continued to be, as long as he lived, an ardent advocate of Swedenborg's teachings. He died suddenly, March 20, 1903, at Palm Beach, Florida, of heart disease, leaving a widow and a daughter, now residing at Denver, Colo., and two sons, Franklin and Norris, both graduates of the Sheffield Scientific School of Yale University—the former now engaged in mining at Leadville, Colo., and the latter just graduated.

Josiah Calef Bartlett was born May 3, 1846, at South Berwick, Maine. He was named after an ancestor who was one of the signers of the Declaration of Independence.

He was educated at Phillips Exeter Academy and graduated from Harvard College in 1869. After serving as instructor in mathematics at Harvard, he became principal of the Bristol Academy at Taunton, Mass. Subsequently he went abroad, took advanced courses in the Freiberg *Bergakademie*, and also studied at Heidelberg. Returning to this country, he became again principal of the Bristol Academy at Taunton, finally retiring to take charge of a manufacturer's mutual insurance association in New England.

In 1888 he became associated with the Chicago, Burlington & Quincy Railway Company, and immediately began investigation of the subject of railway relief departments. When his report was presented to the Board of Directors, the railroad company decided to establish a Relief Department, and Mr. Bartlett was placed at the head of it, continuing in charge until the time of his death.

Mr. Bartlett became a member of the Institute in 1877, and the earlier volumes of its *Transactions* contain valuable contributions from his pen, of which the following is a list:

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	Vol.	Page.	Year.
American Students of Mining in Germany, .	v.	[47], 431	1876.
The Action of Small Spheres of Solids Ascending Currents of Fluids, and in Fluids at Rest,	vi.	[9], 415	1877.

In later years, his engrossing duties in another sphere prevented him from active participation in the work of the Institute; but he retained his membership to the last. He died of cerebral hemorrhage at his home in Chicago, June 1, 1903.

Henry M. Boies, one of the leading citizens of central Pennsylvania, was born in 1837 at Lee, Mass., and died suddenly, December 12, 1903, at the Hotel Sterling, Wilkes-Barre, on his way from Washington to his home at Scranton. The cause of his death was heart-failure, resulting from acute indigestion.

Col. Boies was the descendant of French Huguenot refugees, who came to this country in the seventeenth century and settled in Boston, where they established the first paper-mill in America. He was educated in the New England public schools and in Yale College, from which he was graduated in 1859. The following year he joined and served with the corps of zouaves organized in Chicago by Ellsworth, and when Ellsworth came to New York to form his zouave regiment, Boies aided him as a lieutenant.

Later he became a member of the firm of Silver & Boies, engaged in the freighting and forwarding business at Tivoli, N. Y. He removed to Scranton in 1865 as resident member of the firm of Laflin, Boies & Turck, powder-manufacturers, and four years later became president of the Moosic Powder Company. Noticing that there were many fatal accidents in mines on account of the careless handling of cartridges by lamp-light, and desiring to remedy this evil, in 1873 he invented a cartridge-package for mining-powder which was widely adopted. In 1882 he became president of the Dickson Manufacturing Co., and during the four years he held this position his judicious management was manifest in the increased business of the company, the enlarged works and the introduction of the hydraulic system of flanging and riveting. While with this company he invented a steel-tired car-wheel, now manufactured by the Pressed-Steel Wheel Co., successors to the Boies Steel Wheel Co. He also served as director of other leading manufacturing companies. In the organization of the Third National

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Bank of Scranton he took an active part, and for ten years was a director. In 1886 Governor Beaver appointed him a member of the State Board of Charities, on which he was a member of the Committee on Lunacy and the Executive Committee. He was also identified with the National Prison Association, and as a fruit of his labors in this and other philanthropic societies, published in 1893 a volume, entitled "Prisoners and Paupers," which presents a study of criminality and pauperism, with suggestions for remedy. He also wrote recently an exhaustive work on "The Science of Penology," which has already become an acknowledged text-book on this subject.

Under the administration of Governor Hartranft he became Colonel of the 13th regiment of the Pennsylvania National Guard, which he brought to a high degree of efficiency. But his manifold military, civic, commercial and philanthropic activities did not prevent him from cherishing to the last a cordial relation with this Institute, of which he became a member in 1887.

A widow, two sons and two daughters survive him.

Edwin Arthur Cade was born in 1856 at Ipswich, in the county of Suffolk, England, and educated at the county grammar-school. In early manhood he came to London and was engaged with one of the largest firms of photographers; afterwards, with a firm of manufacturers of photographic supplies, etc.

Marrying the daughter of a merchant trading with West Africa, he formed with his father-in-law a partnership in that business. After some years, a native trader on the Gold Coast having ordered some hammers and picks suitable for gold-mining, Mr. Cade followed the matter up by suggesting that the mines should have some machinery if there was gold enough to warrant it; and when the natives ordered a small plant, Mr. Cade further suggested that the mines could probably be best worked by European capital and management. Having formed a syndicate for preliminary purposes, he proceeded to the Gold Coast and through that British colony, into Ashanti, which was then independent, under King Prempah. He found everything about the mines in the most primitive condition, in the midst of a dense primeval forest, and only

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accessible from the coast by a foot- or hammock-journey of 120 miles, occupying eight days.

He negotiated with the native chiefs for a concession of 100 sq. miles, at the center of which was the mine the Coast natives were then working; and after the usual adventures of West African travel, he returned to England to found a large company. Meanwhile, however, the British Government organized an expedition against the Ashantis and took possession of the country. This caused much delay in further negotiations concerning the concession; but ultimately, Mr. Cade secured a direct title for the property from the Colonial government; and founded the Ashanti Goldfields Corporation, Ltd., with a capital of £250,000, a part only of the capital being issued at first at par, and more recently 20,000 £1 shares having been issued at £25 each. This strong company has vigorously developed the property, and now has a number of mines working, with tram-roads and all necessary buildings and machinery; and a railway has been built by the Government from the Coast to the mines and onward to Kumasi, the capital of Ashanti, under the guaranty of the Ashanti Goldfields Corporation.

Mr. Cade was also a director of the Ashanti Consols, Ltd., a financial and mining company; and the Ashanti Sansu Mine, Ltd., a subsidiary of the Ashanti Goldfields Corporation, formed to work 320 acres of the corporation's property. Finally, there was founded in the autumn of 1902 a new company, the Ashanti Goldfields Auxiliary, of which Mr. Cade was Chairman, and which has very extensive interests in alluvial and quartz properties in Ashanti. It was on business of the last-named company that Mr. Cade was particularly engaged when he contracted the malarial fever which resulted, after twelve days of illness, in his death, May 4, 1903, at Dominasi, in the Gold Coast Colony. He was then 47 years old.

The success attending Mr. Cade's first enterprise in West Africa attracted wide attention, and has resulted in a large number of other companies operating in that sphere, with a special West African share-market on the London stock-exchange.

Mr. Cade had mining interests also in other parts of the world, notably in the Kamloops district, British Columbia,

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which he visited in 1900, when he also traveled in the United States and stayed a little while at New York. He became a member of the Institute in 1903.

Mr. Cade's business career was marked by keen foresight, sound judgment, courageous enterprise, and an exacting attention to the matters on hand. He was highly appreciated for his thorough reliability, and won much esteem by his generous interest in behalf of a large circle of people he came in touch with, as well as of religious and public philanthropic enterprises. He was an able speaker, and a man of large personal influence.

Albert H. Chester was born at Saratoga, N. Y., November 2, 1843; studied in the Columbia School of Mines, N. Y. City, from which he graduated in 1868 with the degree of E.M., receiving from the same institution in 1878 the degree of Ph.D. From 1870 to 1892, he was Professor of Chemistry at Hamilton College, N. Y., and after 1892, Professor of Chemistry and Mineralogy at Rutgers College, N. J. Prof. Chester had a considerable consulting-practice as a mining engineer, but was chiefly known in scientific circles as an authority in mineralogy. He became a member of the Institute in 1871, immediately after its organization; but his only contribution to its *Transactions* was a paper "On the Percentage of Iron in Certain Ores." (*Trans.*, iv., 219.) Though he retained his membership until the time of his death, his professional papers were contributed to chemical or mineralogical societies or periodicals. Prof. Chester was the author of two excellent manuals, a "Dictionary of the Names of Minerals," and a "Catalogue of Minerals with their Chemical Composition and Synonyms." He died from pneumonia, April 13, 1903, in New Brunswick, N. J.

Joseph K. Clark was born in Fayette county, Pa., Feb. 2, 1842. In 1856 he removed to Iowa, and in 1862 to Colorado. A year later he tried his fortune as a gold miner near Santa Fe; and in 1864 he settled at Alder Gulch, Montana. Like other active pioneers of that period, he engaged in various enterprises—store-keeping, transportation, mail-carrying, prospecting and mining. He superintended mines and mills with practical ability, and shared in some of the operations of his

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brother, Senator W. A. Clark. He became a member of this Institute in 1887. Retiring from active business in 1902, he went to reside at Portland, Oregon. He died suddenly Jan. 25, 1904, from apoplexy at Los Angeles, Cal., whither he had gone with his wife to celebrate the birthday of his aged father.

Victor M. Clement was born in San Francisco, Cal., about 1856, and studied at the University of California. Devoting himself to mining, he won while still a young man a high reputation as a skillful manager. During the first reign of terror established in the Cœur d' Alenes, Idaho, by that miners' union which had then, as now, its headquarters at Butte, Montana, and has made the name of the Western Federation of Miners synonomous in at least four States with unspeakable cruelty and outrage, Mr. Clement was superintendent of the Bunker Hill and Sullivan mine at Wardner, and was held as a prisoner for some time by the outlaws, but escaped the torture and death which were inflicted upon so many other victims. Subsequently, he became manager of the Simmer and Jack mines in the Transvaal. Together with Mr. John Hays Hammond, and others, he was sentenced to death by the Boers after the "Jameson raid," and purchased his life and liberty with a large sum of money. After this escape, he became a consulting engineer in West Australia, returning about 1899 to become associated with Capt. J. R. De LaMar in the direction of mining enterprises in Utah, Nevada and California. He joined the Institute in 1887 as a life-member. At the time of his death, he was operating mines near Mazapil, Zacatecas, Mex. He died April 26, 1903, at Saltillo, Mex., from blood-poisoning following a surgical operation. His wife, who survives him, was his constant companion in his many journeys, and was by his side at the end.

Francisco Miguel Coghlan was born September 9, 1852, at the mining town of Catorce, in the State of San Luis Potosi, Mexico. His father was of Anglo-Saxon, and his mother of Indian (with a slight admixture of Spanish) blood. He was left, with his elder brother, to be supported by the latter parent, who sent her sons to the public school of Catorce until poverty obliged her to put them to work, as common laborers, in the San Augustin mine, then, and still partly, owned by the cen-

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tury-old Spanish house of De la Maza, of Catorce, with which, as will be seen, Coghlan's subsequent mining career was exclusively associated.

Soon after, the elder brother was killed by an accident in the mine; and the mother besought of Don Gregorio F. Sedano, superintendent of the Ave Maria mine of the firm above named, to give some less dangerous occupation for her only remaining son. Francisco was accordingly made, in 1867, at the age of 14 years, ore-weigher at the Ave Maria mine. The house of De la Maza was one of the typical old Spanish establishments which conducted farming, banking, etc., as well as mining, and formed the financial back-bone of the Colony, and afterwards of the early Republic of Mexico. Mr. Coghlan was connected with the Ave Maria mine for 16 years, during which period he discharged all functions, above, and underground, becoming Assistant Superintendent, and, in 1883, when his Spanish superior and friend Señor Sedano returned to Spain, the Superintendent of the mine.

The Ave Maria was a small mine and not profitable. But such conditions furnish invaluable stimulus and discipline to genius. Through the friendly aid of an accomplished engineer in Catorce, Coghlan not only learned the theory and practice of surveying, but was also wisely guided in his voracious reading of professional and scientific books. In 1883, when he became Superintendent of the Ave Maria, the house of De la Maza was represented by Don Gregorio, a son of the founder, residing in Spain, and its mining interests in Mexico were managed by Don Vincente Irizar, a Basque, who, entering its service as a youth, fresh from Spain, had become an exceedingly keen practical miner. In 1884, this gentleman purchased the Santa Ana and Anexas properties, which adjoined the Ave Maria, and upon which he had long had his "miner's eye." Don Antonio Hernandez, the seller, an eccentric old man, happened to be godfather to Coghlan, for whom he reserved from the sale a small share in the property. Don Vicente immediately gave the charge of the whole tract, including the Ave Maria and additional areas subsequently secured by purchase or "denunciation," to Coghlan, who personally executed surveys and maps of the entire property.

In 1886, accompanied by Mr. Charles E. Hoffman, an old

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Comstock Lode mining-captain, and Don Blas Escontria, now Governor of the State of San Luis Potosi, Mr. Coghlan visited Nevada and California, to study modern American mining practice; and upon his return, he began the introduction of modern methods and apparatus into the Santa Ana mine. Several later trips to the United States had similar results. For instance, in 1892, after visiting Aspen, Colo., he put into the Santa Ana mine the first electric hoist ever installed in Mexico.

Under his management, the Santa Ana became famous for the millions which it produced, and not less for its bold and extensive exploration and exploitation. The mining work was done chiefly by tunnels; and Mr. Coghlan by skilful training and direction of his workmen in the hard limestone of Catorce, broke all records for speed and economy in this class of mining—equalling in these particulars, by hand-drilling, the results accomplished with power-drills. It was his good fortune to be heartily supported in all his schemes of improvement, and liberally rewarded for his services, by the house of De la Maza, and its Mexican director, Señor Irizar, who frankly attributes to Coghlan's daring and intelligent energy, thus backed, the illustrious record of the Santa Ana.

It would be scarcely possible to estimate too highly the value of the object-lesson thus given to the renascent mining industry of Mexico—a nation which, under the firm and wise direction of its venerable President, with the cordial confidence and support of its people, is so rapidly advancing to the high place for which it is qualified by its natural resources.

Notwithstanding the continuous and arduous labors of his position, Mr. Coghlan managed to make himself a good geologist, mechanic and electrician; to acquire the knowledge of the English, German and French languages; and to gain an acquaintance with medicine which constituted him the most fully trusted physician of the mining population in the midst of which he lived—no matter what other skillful experts in that department might be at hand. He gave to the Santa Ana mine a model hospital and a model school—in fact, it was renowned as a model example of industrial and social organization, and, as such, was visited in 1895, on the invitation of the owners, by President Diaz and a large party of guests.

In 1891 Mr. Coghlan became a member of this Institute,

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and the members who, ten years later, attended the famous Mexican meeting, would have been much gratified to meet the native engineer who had done so much for mining methods in the Republic. But Catorce was too far from the route of their excursion to be visited.

Mr. Coghlan had married, early in life, a lady of Zacatecas, who, with a family of ten children, survives him. In 1900 he resigned the position he had held so long, and devoted the rest of his life to the society of his family, and the management of the large agricultural properties in which he had invested the ample fortune earned by more than thirty years of incessant labor and studies. In this step, he followed the immemorial tradition, according to which Mexicans who have acquired riches by mining become land-owners and enjoy in the relatively quiet occupation of farming, the *otium cum dignitate* of their hearts' desire.

This compliance with a national custom exhibited that peculiar quality of his character and career which most deserves recognition as the secret of his great and beneficent success. Namely, he was, from first to last, an intensely patriotic Mexican. He did not acquire by foreign education conceptions of modern progress which denationalized him, and, returning, attempt to force these new ideas upon a community not prepared for a revolutionary change. On the contrary, being by both blood and breeding a son of the soil, and comprehending sympathetically even those national traits and prejudices which he had outgrown, he selected from other nations, and introduced with skillful tact, those features of progress which would immediately prove themselves beneficial to Mexico and to Mexicans, thus retaining the confidence of his countrymen, while he gradually modified, without rashly uprooting, their methods of thought and action. On a wider scale, we have been recently called to admire the results of a similar transformation, wrought by the statesmen of Japan. But we need not go so far for an illustration. The great leader who has for so many years controlled the progress of Mexico, without losing the national confidence or sacrificing the national traditions, is an instance in point; and to say that Mr. Coghlan, in his sphere of influence, was another Diaz, is to express a deserved and heart-felt compliment to both.

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Personally, Mr. Coghlan was generally taciturn and dignified, though jovial with his intimates, and to his children, who adored him, a comrade as well as teacher; a generous and thoughtful host; a considerate and modest guest; an admirer of talent, and perhaps inclined to overestimate new claimants for such recognition, yet never soured by individual disappointments in this respect; a wise critic of professional propositions; a firm, kind and just administrator of authority; a true friend to all who were under his charge; an earnest and accomplished student of the history of the past, and an intensely interested and well-informed observer of contemporaneous history throughout the world.

His death (from a disease of the liver), which took place January 1, 1903, in the city of San Luis Potosi, terminated a career of such personal merit and public usefulness as to deserve a much more extended notice than can be given here.[1].

Louis Davidson was born in 1854, at London, but came at an early age to Newcastle-on-Tyne, where he was educated at Dr. Bruce's school. He showed exceptional ability in chemistry and physics, and on leaving school, at the age of 14, he was apprenticed to Messrs. Mawson & Swan, manufacturing-chemists, of Newcastle. In this employment he gradually rose until he was offered the post of manager; but preferring to extend his experience, he left this firm and went to London, where he obtained a position in the Brush Electric Light Company; at the same time he joined classes in electrical engineering at the Finsbury Technical College, one of the pioneer institutions in systematic laboratory-work in electricity. He afterwards worked for a time with the firm of Mayfield & Co.; then returned to Newcastle and was for a short time with Messrs. Clarke, Chapman & Co. Next he went to Dublin, where he was manager for a firm of manufacturers of chemical, physical and photographic apparatus. In 1895 he returned to Newcastle and attended classes in geology, mining, mine-surveying, etc., at the Durham College of Science. Becoming a director of the Gowrie & Blockhouse Colliery Company, of Nova Scotia, he went several times to Canada in that capacity, and

[1] For the material used in this sketch, the Secretary is indebted to Mr. Victor M. Braschi, of the City of Mexico.

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superintended the laying out of the pit, and the erection of a wire-rope way for conveying the coals from the pit to the shipping-point. He showed great aptitude for mining-work; and his early death is a great loss to the company. Mr. Davidson was enthusiastic in any study that he took up, and persevering in acquiring a complete knowledge of all its details; and he consequently possessed an extensive knowledge over a wide range of physical and engineering subjects. His circle of friends was not a large one, but those who knew him have to mourn the loss of a true-hearted and constant friend. He died on November 22, 1903, at the age of 49 years, after a brief and painless illness.

William Garrett was born in England in 1844. At the age of 11 he began to work in one of the rolling-mills of Scotland. At 16 he was foreman. Those who remember how exhausting was the labor in a rolling-mill before the introduction of modern machinery for the handling of heavy masses of glowing iron, will appreciate at its true value the fact that young Garrett, deprived of the privilege of attending school, got his education by hard study after his long day's work was done. In almost any other occupation that could be named, this would be easier than in the one he followed. But his genius and ambition were superior to bodily fatigue. In 1878, at the age of 34, he came to the United States, and found employment for five years with the Cleveland Rolling Mill Co. In 1882, he brought out the first form of the deservedly famous Garrett Mill—the first rod-mill in which a No. 5 rod was rolled from a 4-in. (instead of a 2-in., or even smaller) billet, without reheating. On the same principle, the 4-in. billet could be rolled from the ingot without reheating, instead of first reducing the ingot to a bloom, and then reheating the bloom before rolling the billet. The saving thus achieved amounted at that time to about $7 per ton. It is said that about 80 per cent. of the rods now made in the United States are rolled in Garrett mills.

In 1883, Mr. Garrett built at Pittsburg the rod-mill of the Oliver Wire Co.'s South Side plant. In 1886, he built a rod-mill at Cleveland for the American Wire Co. Still later, he became connected with the Joliet Steel Co., and built at Joliet,

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Ill., a rod-mill which is now operated by the American Steel and Wire Co., and is celebrated for its large output.

In 1897, Mr. Garrett returned to Cleveland, where he established, with Messrs. H. W. Lash and John C. Cromwell, the Garrett-Cromwell Engineering Co., which has done much of the new construction called for by large steel companies during the last three years.

Like so many other self-made engineers, Mr. Garrett appreciated the importance of technical education and literature. He was elected a member of this Institute in 1881, and belonged also to other American technical societies, and to the Iron and Steel Institute of Great Britain.

He died July 15, 1903, at Mt. Clemens, Mich., after a stroke of paralysis.

Dana Harmon was born at New Orleans, La., in 1849, and went to California in 1854. He afterwards studied at an Eastern college and returned to California, where he engaged in mining. He was a close observer and investigator, giving particular attention to the milling of gold-ores, concerning which his views were embodied in a book, "The Stamp Milling and Amalgamation of Free Gold-Ores." He became a member of this Institute in 1900.

Mr. Harmon had a wide circle of acquaintances in the mining world. He died at his home in Oakland, Cal., July 9, 1903, of heart disease. A widow and four children survive him.

Frank Arthur Holdsworth was born in 1866 in England and educated at a public school. After acquiring business and professional experience in a civil engineer's office, he left England for South America, where he spent several years in connection with railway-construction and surveys. Finding the climate unfavorable to his health, he gave up this work and took up his residence in California, where he bought a fruit-ranch.

In 1896, Mr. Holdsworth went to the North Island of New Zealand, and was appointed assistant to Mr. H. C. Woolmer, in charge of several gold-mining properties in that district. Somewhat later, as engineer to the Collingwood Gold Estates, Ltd., he executed successfully a scheme for bringing in water for the hydraulic sluicing of that company. He was an asso-

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ciate member of the Institution of Civil Engineers; and in 1899 he became a member of this Institute.

In 1900, he assumed the management of the Royal Oak of Hauraki Company's mine, near Coromandel, N. Z., a post held by him until his death, which came very suddenly in July, 1902, to the great regret of both his employers and his employees, by whom he was held in high esteem.

Benjamin Franklin Jones was born at Clayville, Washington county, Pa., Aug. 8, 1824. He received his early education at New Brighton, Pa., and in 1843 he secured a position with a company operating boats on a canal between Pittsburg and Philadelphia. From 1847 until 1854 (the latter being the year in which the Pennsylvania railroad superseded the old canal-system of freight-transportation) he was a partner in the Mechanics Line of canal-boats.

About the year 1846 he purchased, with Mr. Samuel Kier, his former partner in the canal-boat business, an iron-furnace and forges near Armaugh, Westmoreland county. In 1851 he became connected with the American Iron Works, which, three years later, formed the firm of Jones & Laughlin, a name of great prominence in the history of the development of the iron and steel industries of Pittsburg. This firm subsequently became the Jones & Laughlins, Ltd., and was incorporated in 1903 under the name of the Jones & Laughlin Steel Co. Mr. Jones then retired from active management after fifty years of service.

During the half century that has passed since Mr. Jones became identified with the iron-business in Pittsburg an enormous growth has been attained; and in every step of this progress he was an important factor. Fifty years ago there was not an iron blast-furnace in Allegheny county; and the building of the Eliza furnaces by his company in 1860 marked an era in the introduction of coke blast-furnaces into that district. His firm was among the first to use Lake Superior iron-ores, and it now has important mines in that region. It was also among the first to buy coal-lands in the Connellsville district for making coke. Upon the discovery of natural gas, the company drilled its own wells, and laid its own pipe-lines.

Other instances of its progressive and intelligent enterprise

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might be given, such as its introduction of the famous "cold-rolled shafting," which illustrated a principle in the heat-treatment of steel, the importance of which is now universally recognized in other branches of the manufacture. At the present time, this company employs in its various enterprises nearly 6,000 men, and its high reputation for fair-dealing and thorough work may fairly be attributed in large degree to the traditions established by Mr. Jones.

During half a century of active business management, Mr. Jones endeared himself to his employees by unfailing courtesy and justice. In private life he was universally esteemed. He was married in 1850 to Miss Mary McMasters; and his domestic life was one of undisturbed happiness.

Without being a professional politician or a seeker for office, Mr. Jones always took an active interest in public affairs. In the presidential campaign of 1884, he was chairman of the National Republican Committee.

In 1885, the American Iron and Steel Association elected him its president, to succeed Daniel J. Morrell.

Like many other successful and self-taught business men, to whom the American Institute of Mining Engineers owes an inestimable debt, Mr. Jones appreciated early the value of its endeavor to unite for the exchange of ideas and experiences, the representatives of theory with those of practice. He joined the Institute in 1878, and maintained his connection with it as long as he lived, showing both personally and through his firm a cordial sympathy with its purposes.

He did not long survive his retirement from active business responsibility. His death, which took place at his residence in Allegheny, Pa., May 19, 1903, terminated a long, blameless, upright and useful earthly career.

Julian A. Kebler was born in 1858, at Cincinnati, Ohio. His youth was spent in Massachusetts, and he was graduated in 1878 as a civil engineer from the Institute of Technology in Boston. For a few years he followed this profession, and was in the employ of the Chicago, Burlington & Quincy railroad. He then became connected with J. C. Osgood in the Whitebreast Fuel Co., at Ottumwa, Iowa, where important coal-fields were developed. In 1888 he went to Colorado, and, together

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with J. C. Osgood and A. C. Cass, formed the Colorado Fuel Co., which was consolidated in 1892 with the Colorado Coal & Iron Co., as the Colorado Fuel & Iron Co. Mr. Kebler became second vice-president and general manager, and in 1901 was elected president. In August, 1903, he retired, because of ill-health, from the presidency of the enterprise which his skillful administration had brought to its present great proportions and technical perfection.

Mr. Kebler joined the Institute in 1885, the year of one of its most successful Colorado meetings. On that occasion, and always thereafter, he exhibited cordial good-will and generous hospitality towards its members. He was beloved by his employees as well as his professional colleagues, and was universally recognized as a citizen who had conferred both honor and gain upon his adopted State. He died suddenly, at Denver, Colo., Nov. 20, 1903, from a cerebral hemorrhage. His widow survives him.

Wilbur Clinton Knight was born at Rochelle, Ill., Dec. 13, 1858, and was educated in the common schools of that place. He graduated from the University of Nebraska in 1886. He was Assistant Territorial Geologist of Wyoming in 1886, and from 1888 to 1893 was in charge of mines in Colorado and Wyoming. In 1893 he was elected Professor of Mining Engineering, and in 1894 Professor of Mining and Geology, in the University of Wyoming. This position, together with that of State Geologist of Wyoming, to which he was appointed in 1897, he held until his death, which took place July 28, 1903, at Laramie, Wyoming.

He was a fellow of the Geological Society of America, a member of the American Institute of Mining Engineers (which he joined in 1900), a member of the Geographical Society of America, and the author of many monographs. He was a regular contributor to various technical papers, and his bulletins on "Oils of Wyoming" have been widely quoted and reprinted in America and Europe.

The Smithsonian Institution, the Carnegie Museum, and other institutions possess many specimens, discovered and forwarded by Professor Knight. It was mainly through his efforts that Harvard, Yale and other institutions of learning in

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the East sent expeditions to the Wyoming fossil-fields, where valuable discoveries were made. He was also responsible for the unequaled collection of dinosaurs and other prehistoric animal specimens owned by the University of Wyoming. During a part of 1902 and 1903, he had been on leave of absence from the University of Wyoming, and was connected with the Belgo-American Drilling Trust as consulting chemist.

He was married October 16, 1889, to Miss I. Emma Howell, of Omaha, and leaves one daughter and three sons.

Prof. Knight had done much good work, and his early death is sincerely regretted. His valuable papers on the mineral resources of Wyoming appeared from time to time in the *Engineering and Mining Journal* and *The Mineral Industry*. His years of earnest service in Wyoming especially identified him with that young State, where his loss will be severely felt.

Prof. Knight died from peritonitis, July 28, 1903, at Laramie, Wyo.

Robert Packer Linderman, born at Mauch Chunk, Pa., on July 26, 1863, was the eldest living son of the late Dr. Garrett B. Linderman and his wife, Lucy Evelyn, daughter of the late Judge Asa Packer. His untimely death, on Jan. 21, 1903, at his home on Fountain Hill, South Bethlehem, Pa., was a great shock to all, as it resulted from blood-poisoning caused by an apparently insignificant injury to his hand a few days previously. At the age of 13 Mr. Linderman entered the Mt. Pleasant Military Academy, at Sing Sing on the Hudson, where he spent four years, graduating with the highest honors as valedictorian of his class. In the fall of 1880, after having spent the summer in a tour through Europe, he entered Lehigh University, graduating in 1884, with the degree of Ph.B. For scholarly attainments during his undergraduate course he was elected a member of the Phi Beta Kappa society, and later was for two terms president of the Alumi Association of Lehigh University. He was at the time of his death, and had been for many years past, a member of the Executive Committee of the Board of Trustees of his alma mater.

In the fall of 1884 he entered the employ of Linderman & Skeer, of which firm his father was the senior member. At that time this firm was one of the largest and most important of the individual anthracite coal operators in the State of Penn-

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sylvania. On the death of his father, in September, 1885, he became the head of the firm, and successfully conducted its extensive business until the spring of 1896, when, its coal-beds being practically exhausted, it retired from business.

Prior to this time, however, other and greater responsibilities had been placed upon him. On Jan. 31, 1885, he was elected a director of the Lehigh Valley National Bank of Bethlehem. In the financial field Mr. Linderman proved himself such a ready student that at the death of his father, who was the founder of the institution and the first president, he was elected to the position of vice-president, succeeding the late Francis Weiss, who had been promoted to the presidency. On March 5, 1888, after the death of president Weiss, Mr. Linderman was elected his successor (being at that time, it was thought, the youngest National Bank president in the United States) and continued in this position until his death.

In December, 1885, Mr. Linderman succeeded his father as a director of the Bethlehem Iron Co., and immediately evidenced the deepest interest in its affairs, becoming so conversant with all the intricacies of the business that he was elected vice-president in June, 1888, and president in May, 1890. Thus, at the age of 26, he was at the head of one of the largest and most important of the great iron and steel manufacturing concerns of the world. In the spring of 1899, he was instrumental in organizing the Bethlehem Steel Co., which was capitalized at $15,000,000. This company acquired by lease, for a long period, all the rights, property and franchises of the Bethlehem Iron Co., and Mr. Linderman was elected its first president, while still retaining the same office in the Bethlehem Iron Co. He was president of the Bethlehem Steel Company until August, 1901.

Besides the positions of trust and responsibility outlined above, Mr. Linderman was also a director in the Schuylkill & Lehigh railroad, the Juragua Iron Co., Ltd., Earn Line Steamship Co., Jefferson Coal Co., and American Ordnance Co. Mr. Linderman was also a trustee and a member of the executive committee of Lehigh University, Bishopthorpe School and St. Luke's Hospital, and a vestryman of the Church of the Nativity of South Bethlehem. At the time of his death, he was Burgess of the Borough of Fountain Hill.

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Mr. Linderman joined the American Institute of Mining Engineers in 1890, and was also a member of the Pennsylvania Society of the Sons of the Revolution, the Northampton Club of South Bethlehem, the Metropolitan Museum of Art, the Pennsylvania Society of New York, the University, Lawyers', Engineers', Down-Town, Sigma Phi and New York Yacht clubs of New York, the University, the Art and Manufacturers' Club of Philadelphia and the Metropolitan Club of Washington, D. C. He was a member of the Sigma Phi society and took an active part in the founding of its chapters at Lehigh and Cornell Universities.

On Oct. 15, 1884, Mr. Linderman was married to Ruth May, daughter of Robert H. Sayre, and is survived by five daughters and one son.

George S. Morison was born in New Bedford, Mass., December 19, 1842. He graduated from Harvard University in 1863, and from the Harvard Law School in 1866; but, although admitted to the bar, he never practiced law, preferring to turn his attention to civil engineering, especially to railroads and bridges. He assisted Mr. Octave Chanute in building the bridge across the Missouri river from Kansas City in 1867; in 1871 he became chief engineer of the Detroit, Eel River and Illinois Railroad; from April, 1873, to November, 1875, he was first assistant engineer of the Erie railroad, in which capacity he rebuilt the celebrated viaduct across the Genesee river. In later years he represented the interests of foreign holders as a director in numerous American railroad companies. From 1875 to 1880, he was a member of the firm of Morison, Fields & Co., bridge builders. In 1894, he was appointed by President Cleveland a member of the Board of Engineers to report on the greatest practicable length of a span for the proposed suspension bridge over the Hudson river. In several other important boards of consulting engineers he rendered distinguished service; and was finally in 1898 appointed on the Isthmian Canal Commission by President McKinley. It is generally understood that Mr. Morison's ability and influence were among the chief factors of the change of public opinion to a preference for the Panama route.

During his active career, Mr. Morison built five bridges across

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the Mississippi, and ten across the Missouri river. Among his greatest works was the bridge at Memphis, with its single truss of 790 ft. span, which has been surpassed by only three bridges in the world.

He was a member of the American Society of Civil Engineers, and in 1895 its President. In the American Society of Mechanical Engineers, the Western Society of Engineers, the London Institution of Civil Engineers, the Mexican Society of Engineers and Architects, and other technical and scientific societies he was, also, a distinguished member. He joined the American Institute of Mining Engineers in 1879 and participated occasionally in its discussions; but his professional papers were naturally given to the Societies of Civil and Mechanical Engineers, in the transactions of which more elaborate accounts of his life will be found. The existence and accessibility of these worthier estimates of his work and character constitute the only excuse for the meager nature of this statement concerning so distinguished a representative of American engineering.

Frederick C. Osgood, born at Chicopee Falls, Mass., Jan. 30, 1860, was educated at the Welliston Seminary, East Hampton, Mass., from the classical department of which he was graduated at the age of eighteen. Turning his attention to business pursuits, he entered, in 1883, when only 23 years old, the employ of the Rand Drill Co., as a local salesman attached to the New York office of the Company. At the time of his death, which occurred April 15, 1903, in San Francisco, Cal., he had become, through continuous service and successive well-earned promotions, the general sales-agent of the Company for the United States, superintending all local agencies and visiting frequently all the principal mining districts of the country. Through this experience, utilized by his exceptional quickness of apprehension, good business judgment, honorable character and winning personality, Mr. Osgood became one of the most widely known and highly esteemed expert representatives of the American manufacture of mining machinery. He became a member of the Institute in 1887.

Mr. Osgood was married, Nov. 14, 1901, to Miss Augusta E. Bennett, of Ithaca, Mich., who survives him. It was her sor-

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rowful and precious privilege to be at his bedside when he died far away from home, and to bring his remains across the continent for interment in Chicopee Falls, Mass., by the side of the graves of his parents. The Osgood family, descended from John Osgood, one of the earliest settlers of Andover, Mass., has been distinguished in New England annals by the eminent teachers and divines which have honored its name. Now one of its youngest sons, having exhibited in a different sphere the intelligence, energy and virtues of New England, sleeps with his ancestors, a companion not unworthy of them.

John H. Paddock was born at St. Johnsbury, Vermont, in 1820, and educated in the public school and in the county grammar-school at Montpelier. His father, who had established at Paddock village, on the Passumpsic river, a charcoal blast-furnace and a machine-shop, died in 1845, and the business was continued for many years by the firm of J. C. and J. H. Paddock, the two sons. About 1875, the latter gave up this business, to devote himself to the development of his copper-mine and other properties in Lyman, N. H. During the following eight years, the Lyman mine, under his management, ranked next to the then famous Ely as a producer of copper-ore.

Mr. Paddock was a zealous field-geologist, and thoroughly familiar with every part of northern Vermont and New Hampshire. His invention of the Paddock dry concentrator made him widely known, especially in the West, where the machine was successfully used.

Aside from his vocation as a mining, metallurgical and mechanical engineer, Mr. Paddock was an enthusiastic musician. In his young manhood he was a band-leader, church organist and choir-master, and teacher of the organ and piano. During the Civil War he was aid to the Governor, and active in the care of the troops of his State in the Union army. But all this did not interfere with his keen interest in his profession. He was widely recognized as an expert in mining, and, as such, inspected mines for New England capitalists, traveling on one occasion (about 1880) as far as Mexico.

He might have achieved still higher professional distinction, had not his active career been suddenly cut short by a physical

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calamity. About the beginning of 1884, he fell on the slippery rocks at the Lyman mine and broke his hip. From this injury he never recovered. For twenty years he was almost continuously confined to his house; and during the first months of this trial he was called to bear a more than physical pain in the death of his wife, whose companionship had been his happiness for more than twenty-seven of his years of strength, and was all the more precious in the time of his weakness.

The rest of his life is a simple, yet memorable story of suffering, borne with heroic patience and cheerfulness, and turned to noble uses. He used his enforced leisure in reading, not only the great books of the past, but also in keeping himself fully abreast with the present literature of politics and of scientific and social progress. It is significant that, although he had been a member of this Institute (which he joined in 1882) little more than a year when he was cut off from further professional activity, he continued for twenty years thereafter to pay his dues, in order to aid the cause, and to follow the doings of his more-fortunate colleagues. He was called the best-posted all-round man in his town; and people from distant places, as well as his own townsmen, used to visit him for the pleasure and profit of hearing him talk.

He never forgot the scenes and the people he had visited; and a characteristic feature of his active benevolence, maintained through many years, was the sending of wholesome and useful literature to distant and obscure mining-camps and struggling Southern schools.

He died at St. Johnsbury, March 19, 1903, eighty-three years after his birth in the same town; and at his funeral the bearers were six old men who had worked in his machine-shop more than a generation ago, and some of whom had learned their trade from him.

Augustus E. W. Painter was born in Alleghany, Pa., June 29, 1844. His father, Jacob Painter, of the firm of Zug & Painter, was one of the pioneers in the Pittsburg iron manufacture. He received his education in the academies of that city and at the Rensselaer Polytechnic Institute, at Troy, N. Y. After the death of his father, he became head of the firm, which was afterwards reorganized as the J. Painter & Sons

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Company. Through his successful management of this business, he acquired wealth for his colleagues and himself. As president of a national bank and director in various companies, he exerted a potent influence upon the prosperity of Pittsburg. Himself a trained engineer, Mr. Painter was quick to recognize the value of the Institute, of which he became a member in the first year of its existence. Mr. Painter died July 4, 1904, at Alleghany, Pa.

James Henry Pomeroy was born in Cornwall, England, Nov. 20, 1846, and died (of heart failure) at Portland, Oregon, May 26, 1903. He was a large man, of powerful physique, but when a boy he had suffered from lead-poisoning in the lead-mines of England; and this resulted in a constitutional impairment of the heart.

As a young man he spent several years mining in the tin-, lead-, iron-, and coal-mines of England. In 1872 he came to the United States, settling first in Pennsylvania and removing afterwards to the Pacific Coast. In 1878 he took charge of the mines of the Oregon Iron & Steel Co., at Portland, Oregon, and remained in that position until 1893, when the furnaces of the company were closed.

During later years he engaged, with some associates, in the gold-dredging business in the eastern part of Oregon, and was the first to operate a gold-dredge successfully in the interior of that State.

Thoroughly honorable and reliable, and of a kind and genial disposition, he left many staunch friends. Himself a self-made engineer, he fully appreciated the value of technical knowledge, and had accumulated a considerable library, both scientific and literary. He became a life-member of the Institute in 1889. Mr. Pomeroy left a widow, one daughter and six sons.

Jacob M. Rich was born May 12, 1862, at San Francisco, Cal. After a preliminary education at private schools in New York City, he studied civil and mining engineering at the School of Mines of Columbia University, from which he received the degrees of C.E. and E.M. After this he took a post-graduate course, especially in chemistry, at Johns Hopkins University; and finally, returning to New York, he studied

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medicine at the College of Physicians and Surgeons, receiving the degree of M.D. in 1895. By reason of the failure of his health, shortly after this last graduation, he was never able to employ in active practice the varied scientific and professional equipment which he had spent so many years in acquiring; but he retained to the end his membership in this Institute, which began in 1883, when he was a student of mining. He died March 20, 1903.

August Sahlberg was born in Norway, May 17, 1859. In early life he was a sailor, but in 1879, at the age of 20, he abandoned the sea to seek his fortune in this country. Finally devoting himself to mining as a profession, he worked in the mines of Arizona, New Mexico, Idaho, California and Montana successively, gaining his most important experience and promotion at Butte, Mont. In 1886 he went to Mexico, and, soon after, took charge of what is now the El Oro mine, at El Oro, Mex. Under his management, the enterprise became profitable. He was the first to recognize that the veins running through the El Oro property did not end at its boundary; and he located various claims north and south of the El Oro ground, among which was the now famous Esperanza. This he developed as far as he was able; but, being at that time a poor man, he had not sufficient funds to continue work after he struck water in the shaft he was sinking. After endeavoring to obtain the necessary aid in the United States, he succeeded in securing some local Mexican capital, with which he continued work. The result amply vindicated his judgment. At the time of his death he had a fortune of several million dollars.

Starting with only a common school education, gained in his native land, Mr. Sahlberg acquired by constant application a surprising knowledge of general subjects, besides a thorough practical and theoretical acquaintance with his calling. In addition to his native tongue, he spoke English and Spanish fluently. He was a thorough business man of sterling character, and El Oro owes its present prosperity, in large measure, to his energy and perseverance. He joined the Institute in 1900.

Mr. Sahlberg died April 30, 1903, at his home in Mexico. By his will, after providing for relatives, he divided the greater part of his fortune among old friends and associates.

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Charles Schäffer was born in Philadelphia, Feb. 4, 1838, and was graduated at the age of 21 from the Medical Department of the University of Pennsylvania.

At the beginning of the Civil War, he served as a surgeon in the Union army, but was compelled by ill-health to resign. His arduous duties as a physician, which involved a very large amount of charity work, together with his delicate health, did not permit him to specialize as much as he desired in his favorite study of botany. During the last fourteen years of his life, however, he spent about five months of each year in the mountains of British Columbia, and was thus enabled to make a very complete collection of the flora of that region. At his request, this collection has been presented by his wife to the Academy of Natural Sciences of Philadelphia, which has designated it the Charles Schäffer Rocky Mountain Collection. He took a very active interest in this society, being for several years a member of its Council and Secretary of the Mineralogical and Botanical Sections.

In this age of specialization, with its narrowing influence upon the sphere of the general scientific interests of the individual, it is seldom that one has the opportunity of coming in contact with one who took so intelligent an interest in the many different sciences as did Dr. Schäffer. His many noble qualities as a man, known and appreciated by all who came in contact with him (not the least of these being his innate modesty), are a precious memory to his many friends, both old and young.

Although Dr. Schäffer was not professionally engaged in mining, his interest in the kindred subjects of mineralogy and geology led him to join this Institute as an Associate, in 1882. The broad range of his interest in all scientific subjects, not as a mere dabbler, but as a true student of nature, is reflected in his membership of the following organizations: the American Philosophical Society; the American Geological Society; the Academy of Natural Sciences of Philadelphia; the College of Physicians; the Pennsylvania State Medical Society; the Philadelphia County Medical Society; the American Association for the Advancement of Science; the Franklin Institute; the Geographical Society of Philadelphia; the Photographic Society of Philadelphia; the Historical Society of Pennsylvania; and the Horticultural Society of Pennsylvania.

He died of heart disease, Nov. 23, 1903, at Philadelphia.

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Otis F. Stantial was born at Melrose, Mass., in 1863, and graduated as mining engineer from the Massachusetts Institute of Technology in 1885. Until 1892 he was connected as chemist, successively, with the Friedensville Zinc Co., the North Chicago Rolling-Mill Co., the Crane Co., and the William H. Deering (Harvester) Co., of Chicago. In 1892, he became assistant superintendent of the Illinois Malleable Iron Co., of which he soon became, and remained until his death, general superintendent. Mr. Stantial was an active member of the Western Foundrymen's Association and the American Foundrymen's Association. He joined this Institute in 1900. He died at his home in Chicago, March 7, 1903. His wife and infant son survive him.

Oscar Szontagh was born in 1853, at Vienna, Austria, and received in foreign schools a technical education. He had lived in this country about 20 years, and was widely known in the West as a metallurgist. At one time he was a member of the firm of Szontagh & Costell, San Francisco, Cal. He was the first manager of the smelter at Portland, Oregon, and also laid out a plan of reduction for the Eureka and Excelsior mine, in eastern Oregon. Later, he was engaged in his profession at Salt Lake City and in Colorado. About two years ago he came to Northport, Wash., to take charge of the Le Roi smelter. For the eight months preceding his death, which occurred July 21, 1903, in Spokane, Wash., he had resided in that city.

William J. Taylor was born Jan. 5, 1836, on the old Taylor homestead, at High Bridge, N. J. His early education in the local school was followed by a course at St. Mary's College, Wilmington, Del. In 1856 he engaged in the shipping business at Philadelphia; but ten years later he returned to High Bridge, where he organized the Taylor Iron Co., and built a plant for the manufacture of railway specialties, especially axles and car-wheels, a business which his father managed for many years with energy and ability. In 1876, he removed to Chester, N. J., where he operated several iron-mines and a blast-furnace. Like so many other eastern blast-furnaces, this plant was unable to compete with the great modern establishments, commanding

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vast supplies of cheap ore and fuel, mechanical appliances for handling materials and products, and abundant railway-facilities. The furnace was dismantled in 1891, and Mr. Taylor returned to High Bridge, where he developed the Taylor Iron and Steel Company, and introduced castings of manganese-steel into the United States.* During the late war with Spain, this company furnished many projectiles to the U. S. government.

It was in 1875 that Mr. Taylor became a member of this Institute, of which he was twice (in 1890 and 1899) elected a manager for the term of three years. The following list of his contributions to the *Transactions* indicates the wide range of his practical investigations and suggestive inventions:

	Vol.	Page.	Year.
An Ore-Roasting Furnace,	ix.	304	1881.
A Fluxing Gas-Producer for Making Heating-Gas,	ix.	309	1881.
Experiments with a Straight or No-Bosh Blast-Furnace,	xiii.	489	1884.
The Use of High Explosives in the Blast-Furnace, and of Water-Spray for Cooling in Blowing-Down,	xiii.	670	1885.
Remarks on the Operation of Warwick Furnace, Pa.,	xiv.	861	1886.
A Water-Cooled Gas-Producer,	xv.	822	1887.
Notes on the Energy and Utilzation of Fuel, Solid, Liquid and Gaseous,	xviii.	205	1889.

Mr. Taylor's interest in the Institute was not confined to the professional or scientific sphere. He enjoyed keenly the opportunity for social intercourse afforded by its meetings, which he never failed to attend when he could possibly do so, and his genial companionship will be sorely missed by the survivors (rapidly growing fewer) of that band of veterans who can remember the time when "everybody knew everybody," and the gathering of members was a reunion of friends.

Mr. Taylor was First Vice-President of the Hibbard-Rodman-Ely Safe Co., a director of the Delaware & Bound Brook Railway Co., and an active participant in many other commercial enterprises, especially such as were connected with the introduction of new inventions, towards which his temperament always inclined him. At home, he was known as a public-spirited and liberal citizen. He died Feb. 17, 1903, at his home in Bound Brook, N. J. A widow and five children,

* See *Trans.*, xxiii., 173, 183, 476.

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(together with his father, Mr. Lewis H. Taylor, of High Bridge, who is still active at the age of 92) survive him.

Charles R. L. Tucker was born in London in 1874. He went to Mexico at the age of 17 and was employed as assayer at Zapotlan, State of Jalisco, until 1893; later, until 1895, as assayer at the San Francisco mill in Pachuca, State of Hidalgo. He then went with the Consolidated Kansas City Smelting and Refining Company, first as assayer and then as ore-purchasing agent at Pachuca, Guanacevi, Parral and Jimenez, Mexico. He died August 2, 1902, in St. Luke's Hospital, from a hemorrhage resulting from an attack of typhoid fever.

Robert B. Turner offers another example of death encountered in the heroic endeavor to save the lives of others. He was superintendent in charge of the Kearsarge mine at Virginia City, Montana. A fire broke out in the shaft-house, and the smoke was swept into the mine. The men were just changing shift and all but four got out, when Mr. Turner with three volunteers (their names deserve a record: George Allen, Edward Laherty, and John Hurley) went in to rescue the four. Hurley crawled out barely alive, but the others all succumbed to the smoke. These silent battle-fields demand the highest type of courage, both physical and moral.

Mr. Turner, who was born in Chicago in 1869, was of slender frame and but 34 years old when he died; yet he had already won a high place in his profession. He went to Idaho Springs, Colo., with his parents, as a boy of 13, began practical work as assayer and chemist with Mr. W. N. Wiley, and at 20 years of age went to Revenue Mine, Montana, as assayer. He was soon put in charge of the Amalgamation mill, and, finding the losses heavy, experimented with the cyanide process through the McArthur Forest people in Denver. In two years from his advent (1891) he had erected the Revenue Cyanide mill—the first in Montana, and indeed in the West, except the small testing-mill in Denver. He afterwards got an option on the mill and sold it. Then be began wider operations, erecting and operating a large cyanide plant in Cripple Creek, a concentrating mill in Boulder county, and, with headquarters at Butte, Montana, he designed many small mills, for New Mexico,

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Colorado, Montana and North Carolina, besides examining many mining properties. From 1900 to the time of his death he was engaged in general practice as mining and mechanical engineer at Butte, sometimes taking charge of a mine and erecting the mills—as in 1900 at the Madisonian mine, which he and his brother, W. Frank Turner, bought, inaugurated with milling facilities, and then sold; and the Kearsarge mine, which he began operating in 1902, and for which he was building a mill when, on November 6, 1903, occurred the fatal fire.

Mr. Turner was regarded as an authority on cyaniding. He was offered positions in Mexico, Honduras, the Malay Peninsula and other places, but declined them for domestic reasons. He was a member of the American Society of Civil Engineers, and of the Institute, his membership in the latter dating from 1895.

Frederick A. Wright was born at Copenhagen, N. Y., Oct. 4, 1857. He was actively engaged in mining in the iron- and copper-mines of Michigan for a number of years, but, in 1887, became interested in the development of various mining-camps in other sections. He died of heart-failure at Descubridora, Durango, Mex., Nov. 28, 1903.

NOTE BY THE SECRETARY.—Comments or criticisms upon all papers, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

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OF THE

American Institute of Mining Engineers.

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SECTION I.

INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

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American Institute of Mining Engineers.

(Organized in 1871, and Incorporated in 1905.)

OFFICERS.

For the year ending February, 1906.

Directors.

JAMES GAYLEY (President), R. W. RAYMOND (Secretary),
FRANK LYMAN (Treasurer).

T. A. RICKARD.....New York, N. Y.
CHARLES H. SNOW.....New York, N. Y.
R. W. RAYMOND.....New York, N. Y.
(Terms expire February, 1906.)

JAMES GAYLEY.....New York, N. Y.
FRANK KLEPETKO.....New York, N. Y.
FRANK LYMAN.....New York, N. Y.
(Terms expire February, 1907.)

JAMES DOUGLAS.....New York, N. Y.
JAMES F. KEMP.....New York, N. Y.
ALBERT R. LEDOUX.....New York, N. Y.
(Terms expire February, 1908.)

Council.

PRESIDENT OF THE COUNCIL.

JAMES GAYLEY.

VICE-PRESIDENTS OF THE COUNCIL.

GEORGE W. MAYNARD.....New York, N. Y.
JULIAN KENNEDY.....Pittsburg, Pa.
CHARLES D. WALCOTT.....Washington, D. C.
(Terms expire February, 1906.)

WILLIAM P. BLAKE.....Tucson, Ariz.
THOMAS F. COLE.....Duluth, Minn.
IRVING A. STEARNS.....Wilkes-Barre, Pa.
(Terms expire February, 1907.)

Councilors.

FRANK KLEPETKO.....New York, N. Y.
BENJAMIN B. LAWRENCE.....New York, N. Y.
HEINRICH RIES.....Ithaca, N. Y.
(Terms expire February, 1906.)

F. L. GRAMMER.....Baltimore, Md.
JOSEPH HARTSHORNE.....Pottstown, Pa.
CHARLES H. SNOW.....New York, N. Y.
(Terms expire February, 1907.)

A. A. BLOW.....Denver, Colo.
FRANK LYMAN.....New York, N. Y.
T. A. RICKARD.....New York, N. Y.
(Terms expire February, 1908.)

SECRETARY OF THE COUNCIL.

R. W. RAYMOND.....New York, N. Y.
(Term expires February, 1906.)

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The Incorporation of the Institute.

I. A general meeting of Members and Associates, held December 29, 1904, upon notice duly given according to law, adopted by unanimous vote a resolution authorizing the incorporation of the Institute under the Membership Corporations Act of the State of New York, and with the name of "American Institute of Mining Engineers," by the following persons, to wit: James Douglas, James Gayley, James F. Kemp, Frank Klepetko, Albert R. Ledoux, Frank Lyman, R. W. Raymond, T. A. Rickard and Charles H. Snow, and authorizing the said incorporators to adopt a Constitution and By-Laws for the Institute.

II. This incorporation was completed January 9, 1905; and at a meeting of the incorporators, acting as the Board of Directors, a Constitution and By-Laws were adopted, to be presented for final adoption at the first annual meeting of the corporation. This meeting was held Feb. 21, 1905, the date fixed by the Certificate of Incorporation, which followed in this respect the former Rules.

III. At the said annual meeting, the following Constitution and By-Laws were read in full, and unanimously adopted:

CONSTITUTION.

ARTICLE I.

NAME AND OBJECT.

SEC. 1. This Institute is incorporated under the Membership Corporation Law of the State of New York; its corporate name is AMERICAN INSTITUTE OF MINING ENGINEERS; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: (1) Members; (2) Honorary Members; (3) Associates; and (4) Honorary Associates. Only Members and Associates residing within the United States of America, Republic of Mexico and Dominion of Canada shall be entitled to vote at the meetings of the Institute.

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SEC. 2. All Members, Honorary Members, Associates and Honorary Associates of the American Institute of Mining Engineers as the same existed on the day of the incorporation of this Institute, are Members, Honorary Members, Associates and Honorary Associates, respectively, of this Corporation.

SEC. 3. The following classes of persons shall be eligible for membership in the Institute, namely: as Members and Honorary Members, all professional mining engineers, geologists, metallurgists or chemists, and all persons practically engaged in mining, metallurgy or metallurgical engineering; as Associates and Honorary Associates, all persons desirous of being connected with the Institute who, in the opinion of the Council, are suitable.

SEC. 4. Every candidate for election as a Member or Associate of the Institute must be proposed for election by at least three Members or Associates; must be approved by the Committee on Membership, as prescribed in the By-Laws; and must be elected by the Council. Not less than three-fourths of the votes cast shall be necessary to an election. Every person so elected shall become a Member or Associate, as the case may be, upon payment of his first dues as hereinafter prescribed. Each candidate for Honorary Member or Honorary Associate, must be recommended by at least ten Members or Associates; must be approved by the Council; and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of all the Directors present; provided, however, that the number of Honorary Members and Honorary Associates shall not at any time exceed twenty.

SEC. 5. If any person elected a Member or Associate does not, within sixty days after notice of his election, accept the same and pay his initiation fee and dues for the current year, his election may be cancelled in the discretion of the Council.

SEC. 6. The Council may at any time change the classification of a person elected as an Associate so as to make him a Member, or vice versa. All Members and Associates shall be equally entitled to the privileges of membership, provided that Honorary Members, Honorary Associates, and Members and Associates whose Post-Office addresses shall be outside of the United States, Mexico and Canada, shall not be entitled to vote.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be Ten Dollars per annum, payable in advance on the first day of each Calendar year. Each newly elected Member or Associate shall pay, when notified of election, an initiation fee of Ten Dollars in addition to the dues for the current year. Honorary Members and Honorary Associates shall not be liable to initiation fee or dues. Any Member or Associate in arrears for one year may, at the discretion of the Council, be deprived of the receipt of publications or stricken from the list of Members, provided that he may be restored to membership by the Council on payment of all arrears or may be again proposed and elected after an interval of three years.

SEC. 2. Any Member or Associate not in arrears may become, by the payment of One Hundred and Fifty Dollars at one time, a Life Member or Associate; and shall not be liable thereafter to annual dues.

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ARTICLE IV.

BUSINESS MEETINGS OF THE INSTITUTE.

SEC. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall take place on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented to each annual meeting.

SEC. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint, upon notice to all Members and Associates directed to each at his last known Post-Office address, and mailed in the City of New York not less than twenty days before the date fixed for such meeting.

SEC. 3. At all business meetings of the Institute the presence of nine Members and Associates shall constitute a quorum.

SEC. 4. At all business meetings of the Institute Members and Associates may vote either in person or by proxy, but no Member or Associate in arrears since the last annual meeting shall be entitled to vote.

ARTICLE V.

OTHER MEETINGS OF THE INSTITUTE.

SEC. 1. All meetings of the Institute other than business meetings shall be held at such times and places as the Council may appoint. Notice of all such meetings shall be given to all Members and Associates by mail.

ARTICLE VI.

DIRECTORS AND OFFICERS.

SEC. 1. The business and financial affairs of the Institute shall be managed by a Board of Directors, who shall be elected at the annual meeting in the manner prescribed in the Certificate of Incorporation.

SEC. 2. The officers of the corporation shall be a President, Vice-President, Secretary and Treasurer, who shall be elected by the Directors from among their number. All such officers shall be elected at the first meeting of the Board of Directors after each annual meeting of the corporation, and shall hold office for one year or until their successors are elected and qualify.

The duties of all officers shall be such as usually pertain to their offices, respectively, together with such other duties as may from time to time be prescribed for them by the By-Laws. The Treasurer shall give a bond for the faithful performance of his duties in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

SEC. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation or otherwise, the remaining members of the Board may, by a majority vote, elect a successor to fill the vacancy, who shall continue in office until the next annual meeting or until his successor shall have been chosen.

SEC. 4. The Board of Directors may, in its discretion, declare the place of any Director vacant, on his failure, for any reason, to attend three successive meetings of the Board. Any Director who shall under this section or in any other manner

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cease to be a member of the Board shall, at the same time, be held to have vacated any other office to which he shall previously have been elected; and the Board shall elect a new incumbent to the said vacant office.

SEC. 5. The Board of Directors may from time to time appoint from their own number standing and special committees, and may delegate to such committees such duties as they may see fit.

ARTICLE VII.

MEETINGS OF THE BOARD OF DIRECTORS.

SEC. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held on the third Tuesday in February in each year, after the adjournment of the annual meeting of the Institute.

SEC. 2. Special meetings of the Board of Directors, at which any business may be transacted, may be called to meet at any time at the office of the Institute in the City of New York, by notice in writing mailed at least five days before the meeting, by the Secretary to each member of the Board at his last known Post-Office address, signed either by the President or the Vice-President or by three members of the Board.

SEC. 3. At all meetings of the Board of Directors the presence of five members shall constitute a quorum.

ARTICLE VIII.

THE COUNCIL.

SEC. 1. The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of a Council composed of a President of the Council, six Vice-Presidents of the Council, a Secretary of the Council and nine Councilors, who shall be elected from among the Members and Associates of the Institute in the manner hereinafter prescribed. Members of the Council may or may not be members of the Board of Directors.

SEC. 2. The President of the Council shall be elected for one year, and no person shall be eligible for immediate re-election to this office who shall have held the same for two consecutive years.

After the first year Vice-Presidents of the Council shall be elected to serve for two years, and Councilors shall be elected to serve for three years. No Vice-President of the Council or Councilor shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The Secretary of the Council shall be elected annually.

SEC. 3. At the first annual meeting to be held in the year 1905, there shall be elected a President of the Council to serve for one year, a Secretary of the Council to serve for one year, three Vice-Presidents of the Council to serve for one year, three Vice-Presidents of the Council to serve for two years, three Councilors to serve for one year, three Councilors to serve for two years, and three Councilors to serve for three years. At each subsequent annual meeting there shall be elected a President of the Council to serve for one year; a Secretary of the Council to serve for one year; three Vice-Presidents of the Council to serve for two years; and three Councilors to serve for three years. The term of office of all Members of the Council shall continue until the adjournment of the meeting at which their successors are elected.

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SEC. 4. Vacancies in the Council may occur by death or resignation; or the Council may, by the vote of a majority of all its members, declare the place of any officer or member of the Council vacant, on his failure for one year, from inability or otherwise, to attend the regular meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *provided* that the said appointment shall not render such person ineligible for election to the Council at the next meeting.

SEC. 5. The presence of five Members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or any business coming within the authority of the Council may be transacted at a regularly-called meeting thereof, at which less than a quorum may be present, subject to the approval of a majority of the Council subsequently given in writing to the Secretary and recorded by him with the minutes.

SEC. 6. The election of the Council shall take place at the regular annual meeting of the Institute. Nominations for Members of the Council may be sent in writing to the Secretary accompanied with the names of the proposers at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before said meeting, mail to every Member or Associate entitled to vote a list of all nominations for each office so received, together with the names of the persons ineligible for election to each office; and if the Council or a Committee thereof, appointed for the purpose, shall have recommended any nomination, such recommendation may also be sent to the Members and Associates with the list of all nominations made.

ARTICLE IX.

MEETINGS OF THE COUNCIL.

SEC. 1. Meetings of the Council shall be held at such times and places as the President of the Council or one of the Vice-Presidents of the Council may appoint.

SEC. 2. A meeting of the Council may be held on the day of the annual meeting of the Institute without previous notice. Written notice of all other meetings of the Council, specifying the time and place of such meeting, signed by the Secretary, shall be mailed to every member of the Council at his last known Post-Office address at least ten days before the date of the meeting.

ARTICLE X.

PAPERS AND PUBLICATIONS.

SEC. 1. The Council shall have power to decide as to the acceptance and publication of any professional papers presented to the Institute, subject to such conditions as the Board of Directors may prescribe.

SEC. 2. The copyright of all professional papers communicated to and accepted by the Institute shall be vested in it, unless otherwise expressly agreed between the Council and the author. The Institute shall not assume responsibility for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Council nor the Institute shall officially approve or disapprove any technical or scientific opinion or any proposed enterprise, outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

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SEC. 3. Special Committees may from time to time be appointed by the Council to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE XI.

SUSPENSIONS AND EXPULSIONS.

SEC. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

SEC. 2. Any member of the Institute may be suspended or expelled for misconduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared by the Council and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused member notice thereof in writing, mailed to him at his last known Post-Office address not less than thirty days before said date, accompanied by a full copy of the charges and a copy of the second, third and fourth sections of this article.

SEC. 3. Upon the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative; hear any witnesses who may be called in support of the charges and at his option cross-examine the same; and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defence, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

SEC. 4. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defence, the Board shall fix a date for a rehearing within three months from the receipt of such affidavit and shall immediately notify the accused member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defence as he would have had upon the original hearing; and after the defence is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

SEC. 5. All interests in the property of the Institute of persons resigning, or otherwise ceasing to be Members or Associates, shall vest in the Institute.

ARTICLE XII.

AMENDMENTS.

SEC. 1. This Constitution or any Article or Section thereof may be amended at any annual meeting by a two-thirds vote of all the members present in person or by proxy, *provided* that notice of the proposed amendment shall have been given

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in writing at a previous meeting, and *provided also* that the amendment or amendments so adopted shall have been printed and mailed to all Members and Associates not later than thirty days before the annual meeting. Any amendment or amendments approved by a majority of the votes cast shall be deemed to have been adopted, and shall become a part of this Constitution. The Secretary shall forthwith print and distribute to Members and Associates an announcement of the result of said vote, and if any amendment or amendments shall have been adopted, a copy of the section or sections so amended.

BY-LAWS.

I. PRESIDING OFFICERS.

At all Business meetings of the Institute the President, or, in his absence, the Vice-President, or, in the absence of both of them, any other member of the Board of Directors to be chosen by the meeting, shall preside.

At all other meetings of the Institute the President of the Council or, in his absence, one of the Vice-Presidents, if present, shall preside.

II. ORDER OF BUSINESS.

At each Business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Election of Members of the Council.
7. Reports of Standing Committees.
8. Reports of Special Committees.
9. Special Orders.
10. Miscellaneous business.

This order of business may be changed by a vote of a majority of the Members and Associates present in person or by proxy.

The usual parliamentary rules shall govern all meetings of the Institute except in cases otherwise provided by the Constitution or the By-Laws.

At all sessions of the Institute other than business meetings, the order of proceedings, and the time of adjournment, shall rest in the discretion of the presiding officer.

III. SECRETARY.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the Corporate Seal, of the Minute Books, and of all Legal Documents belonging to the Institute. He shall conduct, on behalf of the Institute, all correspondence relating to business matters, except such as pertains directly to the office of the Treasurer.

He shall notify all officers and Directors and Members of the Council, and all Members of Committees of their election and appointment; shall issue notices of

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all meetings of the Board, and of the annual and other meetings of the Institute; and shall, in calling special meetings of the Directors, specify the object of such meeting.

IV. SECRETARY OF THE COUNCIL.

The Secretary of the Council shall act as the Clerk of that body at all of its meetings and at all meetings of the Institute called for the discussion of professional, technical or scientific matters, or for any other purpose than the transaction of business.

He shall be custodian of all technical or scientific papers submitted to the Institute for its consideration, shall have charge of the editing and printing of all material published by the Institute, and of the distribution thereof. On the first day of May following the year in which each volume of *Transactions* is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him. He shall have charge of all the correspondence of the Institute relating to other than business affairs.

The Secretary of the Council shall receive a salary to be fixed by the Board of Directors. He may appoint an Assistant Editor, who shall likewise receive a salary to be fixed by the Board of Directors.

The Secretary of the Council may or may not be the same person as the Secretary of the Institute.

V. ASSISTANT SECRETARY.

The Secretary may, with the approval of the Board of Directors, appoint an Assistant to whom both he and the Secretary of the Council may delegate such of his or their duties as he or they may see fit. This Assistant Secretary shall receive such salary as shall be fixed by the Board of Directors, which shall cover his services both to the Secretary and to the Secretary of the Council.

VI. TREASURER.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute and at every meeting of the Board of Directors at which such report shall be called for, the balance of money on hand, and any existing appropriation which may affect the same.

His accounts shall be audited annually by a Committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which Committee shall report thereon at such annual meeting.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding $5,000, in a special account in a Bank or Trust Company, subject to the draft of the Assistant Treasurer, and may delegate to the Assistant Treasurer the duty of paying, out of this account, the current expenses of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to the Assistant Treasurer or not.

VII. ASSISTANT TREASURER.

The Treasurer may appoint, with the approval of the Board of Directors, an Assistant Treasurer, to whom he may delegate the duty of conducting the correspondence incidental to the office of Treasurer, of receiving and depositing in

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bank to the credit of the Institute all moneys received, and of paying, out of the special account upon which he may be authorized to draw, the necessary expenses of the Institute. The Treasurer may require of him a bond, running to the Treasurer personally, in an amount not exceeding $5,000, the expense of which shall be borne by the Institute.

The Assistant Treasurer shall receive such compensation as shall be fixed by the Board of Directors.

The offices of the Assistant Secretary and of the Assistant Treasurer may, if so desired by both the Secretary and the Treasurer and approved by the Board of Directors, be united in the same person, who shall then receive the salary of both offices.

The Assistant Treasurer may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force for himself, the Assistant Secretary and the Secretary of the Council, at such salaries as shall be approved by the Board of Directors. He shall, if the offices of Assistant Secretary and Assistant Treasurer be united in the same person, be the immediate superior of all such employees, unless the Secretary of the Council or the Treasurer be present, in which event either of them shall be the superior of all employees, including their respective assistants.

VIII. STANDING COMMITTEES.

The Standing Committees of the Institute shall be three in number, known respectively as the FINANCE COMMITTEE, the LIBRARY COMMITTEE and the COMMITTEE ON MEMBERSHIP.

The FINANCE COMMITTEE and the LIBRARY COMMITTEE shall each consist of three members of the Board of Directors, and shall be appointed by the President at the first meeting of the Board, after the annual meeting in each year.

The COMMITTEE ON MEMBERSHIP shall consist of five Members of the Council, and shall be appointed by the President of the Council, at the first meeting of the Council after the first annual meeting in each year.

IX. FINANCE COMMITTEE.

It shall be the duty of the FINANCE COMMITTEE to inquire into and examine the financial condition of the Institute, and to consider ways and means of increasing its revenue and of limiting its expenses. It shall report from time to time to the Board as often as it may deem expedient, and whenever it shall be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in every year, examine the securities belonging to the Institute in the custody of the Treasurer, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the FINANCE COMMITTEE, but shall attend the meetings of the same if requested to do so.

X. LIBRARY COMMITTEE.

The LIBRARY COMMITTEE shall be the custodian of all books in the Institute Library and of additions thereto; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May, of each year, receive from the

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Secretary of the Council, and receipt for same to him, all the volumes of *Transactions* for the preceding year, not then distributed by said Secretary.

It shall cause to be kept, under the direction of the Assistant Secretary, a catalogue of all books in the Library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged to it all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

The receipts from the sale of any volume of *Transactions* taken from the custody of the LIBRARY COMMITTEE shall be credited to the LIBRARY COMMITTEE on the books of the Treasurer, and devoted to the general purposes of the Institute.

XI. COMMITTEE ON MEMBERSHIP.

All nominations for Members or Associates of the Institute shall be submitted to and passed upon by the COMMITTEE ON MEMBERSHIP, who shall report thereon to the Council. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualifications of each one. Its proceedings shall be secret and confidential.

No member of the Committee shall propose any candidate.

XII. ELECTION OF MEMBERS.

After the COMMITTEE ON MEMBERSHIP shall have reported to the Council its conclusions as to the acceptability of each candidate, the Council shall vote upon the same.

Two negative votes of members of the Council present shall prevent the election of any candidate. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Council.

XIII. UNITED ENGINEERING SOCIETY.

The Board of Directors shall, at its first meeting after the adoption of these By-Laws, designate three Members or Associates of this Institute to be representatives of this Institute upon the Board of Trustees of the UNITED ENGINEERING SOCIETY, making at the same time provision for the expiration of the terms of office of said representatives, as provided in the By-Laws of the said UNITED ENGINEERING SOCIETY.

At the last meeting of the Board of Directors prior to the first day of each January thereafter, the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said UNITED ENGINEERING SOCIETY for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute in the Board of Trustees of said Society, by reason of the death, resignation or removal of any such representative therein, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

XIV. AMENDMENTS.

These By-Laws may at any time be altered or amended by a vote of three-fourths of the Board of Directors, or by the Members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

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IV. After the adoption of the Constitution and By-Laws, the meeting proceeded to the election of officers by vote of the members and associates present in person or by proxy; and the following were unanimously elected:

DIRECTORS.

For one year, T. A. Rickard, Charles H. Snow and R. W. Raymond;
For two years, James Gayley, Frank Klepetko and Frank Lyman;
For three years, James Douglas, James F. Kemp and Albert R. Ledoux.

COUNCIL.

PRESIDENT OF COUNCIL.

JAMES GAYLEY, ...	New York, N. Y.

VICE-PRESIDENTS OF COUNCIL.

(To serve one year.)

GEORGE W. MAYNARD, ...	New York, N. Y.
JULIAN KENNEDY, ...	Pittsburg, Pa.
CHARLES D. WALCOTT, ...	Washington, D. C.

(To serve two years.)

WILLIAM P. BLAKE, ...	Tucson, Ariz.
THOMAS F. COLE, ...	Duluth, Minn.
IRVING A. STEARNS, ...	Wilkes-Barre, Pa.

SECRETARY OF COUNCIL.

R. W. RAYMOND, ...	New York, N. Y.

(To serve one year.)

COUNCILORS.

(To serve one year.)

FRANK KLEPETKO, ...	New York, N. Y.
BENJAMIN B. LAWRENCE, ...	New York, N. Y.
HEINRICH RIES, ...	Ithaca, N. Y.

(To serve two years.)

F. L. GRAMMER, ...	Baltimore, Md.
JOSEPH HARTSHORNE, ...	Pottstown, Pa.
CHARLES H. SNOW, ...	New York, N. Y.

(To serve three years.)

A. A. BLOW, ...	Denver, Colo.
FRANK LYMAN, ...	New York, N. Y.
T. A. RICKARD, ...	New York, N. Y.

V. At a meeting of the Board of Directors, held February 21, 1905, after the adjournment of the annual meeting, the following officers were elected: *President*, James Gayley; *Secretary*, R. W. Raymond; *Treasurer*, Frank Lyman.

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ANNUAL REPORT.

At the Annual Meeting of the AMERICAN INSTITUTE OF MINING ENGINEERS, held on the 21st day of February, 1905, at 99 John Street, on motion duly made and seconded, the Secretary was instructed to publish, for the information of members, a statement of the receipts and expenditures of the unincorporated American Institute of Mining Engineers, for the year 1904, and to the 9th day of January, 1905, the date of the incorporation of this Institute.

In pursuance of the above instructions, the Secretary publishes the statement of receipts and payments for the above period, duly certified by Barrow, Wade, Guthrie & Company, certified public accountants (see pages xviii and xix).

Changes in membership have taken place during the year as follows: 276 members and 29 associates have been elected; 19 associates have become members; 3 members have been reinstated in payment of arrears of dues, coupled with satisfactory explanation of previous default; 2 honorary members, 37 members and 2 associates have died; 39 members and 8 associates have resigned; and 45 members have been dropped for nonpayment of dues. These changes are tabulated as follows:

Membership of the American Institute of Mining Engineers, January 1, 1905.

	H. M.	M.	A.	Totals.
Membership Dec. 31, 1903	9	3,306	190	3,505
Gains: By Election	.....	276	29	305
Change of Status	.....	19	.....	19
Reinstatement	.....	3	.....	3
Losses: By Resignation	.....	39	8	47
Dropping	.....	45	.....	45
Change of Status	.....	.....	19	19
Death	2	37	2	41
Total gains	.....	298	29	327
Total losses	2	121	29	152
Membership Dec. 31, 1904	7	3,483	190	3,680

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The list of deaths reported during the year 1904 comprises the following names (the figures in parenthesis indicate the year in which the persons named were elected to membership).

MEMBERS DECEASED, 1904.

Honorary Members.—Sir Lowthian Bell (1897), Dr. Thomas Messenger Drown (1871: Honorary, 1883).

Members and Associates.—Jacob Neff Barr (1884), A. P. Blandy (1903), A. Cass Canfield (1876), William E. Carroll (1901), *William E. C. Coxe (1874), Russell W. Davenport (1877), *Isidor Davidov (1893), †Louis Davidson (1889), Louis M. Davis (1884), †Frederick W. Dobler (1902), Edward Doerr (1897), Louis C. DuBois (1900), Edward M. Fergusen (1878), C. Le Neve Foster (1893), George Clinton Gardner (1878), William J. Harris (1871), H. D. Hoskold (1894), William S. Hungerford (1882), Edward P. Landis (1882), August Lengemann (1902), William M. Lippitt (1900), Robert J. McCartney (1897), Henry McCalley (1888), A. McClellan (1871), H. G. Morse (1881), Arthur S. Mahony (1888), Philip W. Moen (1888), Clarence V. Page (1900), Charles Parkin (1875), Rubens P. Patterson (1886), Henry Paul (1889), William H. Pettee (1871), James B. Randol (1879), Irving W. Sevier (1904), William J. Sims (1902), John H. Slavens (1901), *Charles B. Squier (1886), Charles W. Stickney (1891), Homer H. Swaney (1903), Oliver Williams (1887), Whittaker Wright (1881), Walter B. Wilson (1892).

Of these, Sir Lowthian Bell, Dr. Thomas Messenger Drown, Sir Clement Le Neve Foster, and Prof. William H. Pettee will be made the subject of special Biographical Notices, to be separately published.

[SECRETARY'S NOTE.—Since it has been impossible to complete these obituary notices before the date of the annual meeting, the said notices will be printed and distributed hereafter in a separate pamphlet, and subsequently published in Vol. xxxv. of the *Transactions* in their proper place as the concluding portion of the Proceedings of the Annual Meeting.—R. W. R.]

* Death not reported until after January 1, 1905, and, therefore, not included in the table on p. xvi.

† Death occurred in 1903, but not reported until after January 1, 1904, and therefore included in the table on p. xvi.

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STATEMENT OF RECEIPTS AND PAYMENTS FOR THE

RECEIPTS.

1904.				
Jan. 1.	To Balance forward from last year,			$6,625 65
1905.				
Jan. 9.	To Dues, etc.:			
	Annual Dues,	$38,087 37		
	Life Memberships,	1,250 00		
	Initiation Fees,	1,030 30		
			$40,367 67	
	To Binding of *Transactions*,	3,650 33		
	Sale of Publications,	3,546 87		
	Electrotypes,	166 30		
	Miscellaneous,	387 88	7,751 38	
				48,119 05
	To Interest received:			
	Pennsylvania Railroad $13,000 General Mortgage Bonds at 6 per cent.,		780 00	
	United States Government $2,900 4 per cent. Bonds,		29 00	
	New York Central & Hudson River R. R. Lake Shore Collateral $6,000 3½ per cent. Bonds,		210 00	
	Bank Deposits,		261 72	
				1,280 72
	To Reimbursement from Library Fund for Purchase of Books,			2,051 56
	To Sale of Bonds:			
	United States Government $2,900 4 per cent. Bonds,			3,129 28
				$61,206 26
1905.				
Jan. 9.	To Balance represented by Cash in Banks and on hand,			$9,401 58

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PERIOD JANUARY 1, 1904, TO JANUARY 9, 1905.

PAYMENTS.

1905.			
Jan. 9.	By Expenditures for the year:		
	Printing Volume xxxiii. of the *Transactions*,	$1,729 15	
	Printing Volume xxxiv. of the *Transactions*,	4,070 19	
	Printing Pamphlet Edition of Papers,	6,686 21	
	Printing Circulars and Ballots,	512 75	
	Binding Volume xxxiii. of the *Transactions*,	2,485 00	
	Binding Volume xxxiv. of the *Transactions*,	2,735 37	
	Binding Miscellaneous Volumes,	427 95	
	Binding of Exchanges,	191 40	
	Engraving and Electrotyping,	1,211 53	
	By Editorial Appropriation and Salaries:		
	Secretary's department, including Clerks, Stenographers and expenses of proofreading and editing,	8,839 26	
	Assistant Secretary's and Assistant Treasurer's department,	6,757 73	
	Librarian and Assistant,	1,281 00	
	Postage, including Post Office Box rental,	3,741 96	
	Stationery,	733 23	
	Rent,	2,708 33	
	Express and Freight Charges,	3,147 87	
	Telephone,	289 39	
	Telegrams, Cables and Carfare,	85 24	
	Storage of *Transactions*,	154 20	
	Office Supplies and Repairs,	325 89	
	Refunding Payments,	16 26	
	Insurance Premiums, Fire and Surety,	207 75	
	Collection Charges,	24 78	
	Extra Clerical help,	239 65	
	Special Stenographers and expenses of Meetings,	1,122 28	
	Auditing,	75 00	
	Incorporation Expenses,	2 00	
			$49,801 37
	By Office Equipment,	333 52	
	Library Additions,	1,669 79	
			2,003 31
	By Balance,		9,401 58
			$61,206 26

NEW YORK, N. Y., February 17, 1905.

We have examined the above statement, compared it with the books and vouchers and find same correct. In addition to the balance of Cash in Banks and on hand of $9,401.58, the Institute own Investments of a par value of $19,000, consisting of thirteen (13) Registered 6 per cent. Bonds of The Pennsylvania Railroad Company, $13,000, and six (6) 3½ per cent. Bonds of the New York Central and Hudson River Railroad Co. Lake Shore Collateral, $6,000. These Securities have been examined by us and found to be in order.

(Signed) BARROW, WADE, GUTHRIE & Co.,
Certified Public Accountants.

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MEMBERSHIP.

The following list comprises those persons elected as members who duly accepted election in January and February, 1905.

Cuyler Adams, Mine Owner, ... Duluth, Minn.
Edgar C. Agabeg, Min. Engr., Panuria, via Sitarampore, E.I.R., Bengal, India.
Rupert C. Alabaster, Met., 28 Dawson Place, Bayswater, London, Eng.
Peter T. Austen, Chem., ... 89 Pine St., N. Y. City.
Melbourne Bailey, Gen. Mgr., ... Stanley, B. C., Canada.
Karl Baumgarten, Assayer, Cerro Prieto Mine, Magdalena, Sonora, Mex.
Joseph W. Bible, Mine Mgr. and Engr., ... Hanover, N. M.
Robert Montgomery Bird, Supt. Treatment Dept.,
So. Bethlehem Steel Co., So. Bethlehem, Pa.
Edward N. Breitung, Mine Mgr., Savings' Bank Bldg., Marquette, Mich.
Wallace B. Bretherton, Chem., ... Val Verde, Ariz.
Raymond Brooks, Asst. Supt., Chimney Rock, Park Co., Mont.
Harry H. Burhaus, Mine Surveyor, ... San Fernando, Durango, Mex.
Arthur Howe Carpenter, Met. and Min. Engr., 1420 Josephine St., Denver, Colo.
David Cole, Min. and Mech. Engr.,
Cananea Cons. Copper Co., Cananea, Sonora, Mex.
John P. Collins, Blast-Furnace Supt., 222 Mathilda St., Pittsburg, Pa.
Walter H. Crawford, Min. Engr., 218 Union St., Nashville, Tenn.
Charles T. Fairbairn, Mine Supt., ... Virginia, Minn.
Oscar Falkman, Min. Engr., ... Box 491, Donora, Pa.
Walter Fitch, Mine Mgr., ... Salt Lake City, Utah.
T. F. M. Fitzgerald, Mine Supt., ... Flat River, Mo.
Alfred Fox, Jr., Gen. Mgr. and Chief Engr., . . Avequipa, Peru.
John A. Garcia, Min. Engr., 3109 Grand Ave., St. Louis, Mo.
Alexander S. Garfield, Cons. Engr., ... 44 Broad St., N. Y. City.
H. W. C. Gender, Mine Mgr., Okanagan Landing, B. C., Canada.
Lewis H. Greene, Min. Engr., ... Los Angeles, Cal.
Charles H. Halcomb, Steel Mfr., ... Syracuse, N. Y.
William J. Hamilton, Met., ... Aguascalientes, Mex.
Joseph H. Hutchinson, Min. Engr., ... Boise, Idaho.
James Duane Ireland, Min. Engr., ... Camp Bird, Ouray Co., Colo.
Fred L. Johnston, ... Berkeley, Cal.
Owen Jones, Min. Engr., ... Arequipa, Peru.
Fred A. Jordan, Instructor, Michigan College of Mines, Houghton, Mich.
Carl E. Julihn, Min. Engr., 4159 Magnolia Ave., St. Louis, Mo.
Harry L. Kaufman, Mine Mgr., ... Marquette, Mich.
Charles Kirchen, Met., ... Sumpter, Oregon.
Ralph Crane Kline, Chem. and Met., ... Lajolla, San Diego Co., Cal.
Watkin W. Kneath, Portland Cement, 129 S. 4th St., Salt Lake City, Utah.
Maurice Lindeman, Met., ... De Lamar, Nev.
J. M. McClave, Met. Engr., 1734 Arapahoe St., Denver, Colo.
John McGee, Min. Engr., ... Kendall, Mont.

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Mark G. Magnuson, Assayer, ... Sonora, Mex.
F. A. G. Maxwell, Min. Engr., ... Randfontein, Transvaal, S. A.
Lucius W. Mayer, Min. Engr., ... Victor, Colo.
Harry T. Mercer, Min. and Civ. Engr., ... Trimountain, Mich.
Thomas H. O'Brien, Mine Supt., ... Fierro, N. M.
Edwin N. Ohl, Peoples' Savings' Bank Building, Pittsburg, Pa.
Robert S. Oliver, Min. Engr., ... Trimountain, Mich.
Lewis E. Paddock, Min. Engr., ... 18 West 3d St., Duluth, Minn.
Joseph E. Parker, Mech. and Min. Engr.,
P. O. Box 213, Bulawayo, Rhodesia, S. A.
Talbot E. Pierce, Semet Solvay Co., Syracuse, N. Y.
Harold Playter, Mine Supt., ... Goldboro, N. S., Canada.
Davis Richardson, Mine Owner, 709 Lankershim Bldg., Los Angeles, Cal.
H. F. A. Riebling, Cons. Engr., ... Cripple Creek, Colo.
John J. Rutledge, Min. Engr. and Geol., 213 Courtland St., Baltimore, Md.
William J. Seaton, Engr., ... Rosebank, Staten Island, N. Y.
Carlos Sellerier, Min. Engr., ... Mexico City, Mex.
Howard I. Shaw, Supt., ... Kendall, Mont.
Thomas Henry Sheldon, Paymaster Portland Gold Min. Co., Victor, Colo.
Charles Harold Showers, Mine Owner, ... Colima, State of Colima, Mex.
Tomas Solomon, Mine Supt., ... Apartado 13, Pachuca, Mex.
Harry Stansfield, Mine Engr. and Mine Mgr., . . Auckland, New Zealand.
Clarles C. Starr, Min. Engr., ... Wardner, Idaho.
John W. Stull, Mine Supt., ... Lignite, Va.
Kuchiro Takagi, Mine Supt., Mitsen Muke Coal Mines, Chikuago, Japan.
Edward T. Temby, Civ. and Min. Engr., Francistown, Matabeleland, S. A.
Edward T. Thornton, Mine Supt., ... Velardena, Durango, Mexico.
Bruce Walter, Engr., ... 323 Evaline St., Pittsburg, Pa.
George Wilcox, Min. Engr., Franklin Furnace, Sussex Co., N. J.
Edwin C. Witherby, Gen. Supt., Semet Solvay Co., Syracuse, N. Y.
Samuel S. Wyer, Mech. Engr., 1210 Highland St., Columbus, Ohio.

ASSOCIATES.

Lawrence M. Buck, Student, 1228 St. Paul St., Baltimore, Md.
Thomas Truman Fulton, Mine Mgr., ... Bass River, N. S., Canada.
Ambrose E. Ring, Min. Engr., Sheridan Ave. and 159th St., N. Y. City.
Benjamin F. Tillson, Student, 132 Wall St., New Haven, Conn.

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Bi-Monthly Bulletin.

For the convenience of persons who desire to file, or otherwise use separately, the technical papers in Section II. of the Bulletin, each of these papers has been paged and wired by itself; the whole collection being held together by a single, heavy wire, upon the removal of which it will fall apart into individual pamphlets, substantially like those formerly issued.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

All communications concerning the contents of this Bulletin should be addressed to R. W. Raymond, Secretary, 99 John St., New York City (P. O. Box 223; Telephone number 5477 John).

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LIBRARY.

The library has a complete set of the *Journal of the Iron and Steel Institute*, excepting pages 1–276 of volume 1871. As this periodical has proved to be of inestimable value to persons consulting it, the Institute is very desirous of supplying the pages now lacking. Members are asked to coöperate with the library in securing this part of volume 1871.

Contributions of books and pamphlets are respectfully solicited from members, especially from those who may have duplicate copies of works on subjects of interest to Mining Engineers.

Accessions.

December 16, 1904, to February 15, 1905.

DONORS AND DONATIONS.

FROM THE ACADÉMIE ROYALE DE BELGIQUE.

Bulletin de la Classe des Sciences. Nos. 1–8, 8°, Bruxelles, 1904.

" R. ÅKERMAN.

SUNDBARG, G. *Sweden, Its People and Its Industry*, xi + 1141 p. il. pl., 8°, Stockholm, 1904.

" THE AMERICAN CERAMIC SOCIETY.

Transactions, vol. vi., 8°, Columbus, 1904.

" THE ART CLUB OF PHILADELPHIA.

Charter, Constitution, By-Laws and House Rules, with List of Members, 12°, Philadelphia, 1904.

" R. BECK.

Über die Erzlager der Umgebung von Schwarzenberg im Erzgebirge, Part 2, il., 8°, Freiberg in Sachsen, 1904.

" W. B. BARBER.

On the Lamprophyres and Associated Igneous Rocks of the Rossland Mining District, British Columbia, 8°, Minneapolis, 1904.

" THE BLACK HILLS MINING MEN'S ASSOCIATION.

Papers Read Before the Black Hills Mining Men's Association . . . on Mining and Metallurgy of Black Hills Ores . . ., 135 p., il., 8° [Omaha], 1904.

" THE BOSTON PUBLIC LIBRARY.

Annual List of New and Important Books Added to the Public Library, 1903–'04, 8°, Boston, 1905.

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FROM B. H. BROUGH.

BROUGH, B. H. *Historical Sketch of the First Institution of Mining Engineers*, 13 p., 8°, London, 1899.

" THE BUREAU OF MINES, ONTARIO.

ONTARIO, BUREAU OF MINES. *Report*, 13, 8°, Toronto, 1904.

" THE BUREAU OF THE MINT.

U. S., BUREAU OF THE MINT. *Annual Report of the Director*, 1904, 8°, Washington, 1904.

——— *Report upon the Production of Precious Metals*, 1903, 8°, Washington, 1904.

" CARNEGIE INSTITUTE, PITTSBURGH.

Ninth Celebration of Founder's Day, November 3, 1904, 8°, Pittsburg, 1904.

" CARNEGIE LIBRARY, PITTSBURGH.

PITTSBURGH, CARNEGIE LIBRARY. *Alphabetical Finding List of Periodicals and Other Serials Received by the Carnegie Library of Pittsburgh*, ed. 4, 21 p., 8°, Pittsburg, 1904.

——— *Bibliography of Steam Turbines*, 23 p., 8°, Pittsburg, 1904.

——— *List of the Publications of Scientific Societies and the Periodicals on Pure and Applied Science in the Reference Department of the Carnegie Library of Pittsburgh*, 19 p., 8°, Pittsburg, 1900.

——— Monthly Bulletin, vol. ix., Nos. 4 and 6, 8°, Pittsburg, 1904.

" CASE SCHOOL OF APPLIED SCIENCE.

Catalogue of Case School of Applied Science, 1904–'05, 12°, Cleveland, 1905.

" COLUMBIA UNIVERSITY.

COLUMBIA UNIVERSITY SCHOOL OF MINES. *Announcement of Courses in Mining Engineering and in Metallurgy*, 1904–'06, ed. 2.

" CORNELL UNIVERSITY LIBRARY.

Librarian's Report, 1903–'04, 16°, Ithaca, 1905.

" CHROME STEEL WORKS, NEW JERSEY.

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SECTION II.

TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents, pages i and ii.

ERRATA.

Bi-Monthly Bulletin No. 1, January, 1905.

The following corrections of the papers contained in Section II. of the Bi-Monthly Bulletin No. 1, January, 1905, have been received and will be made before the papers are printed in the bound volume of the *Transactions*. Said corrections are here published in order to bring them promptly to the notice of the Institute.

Page 108, thirteenth line from bottom, for "2,900° to 2,200° F." read "1,900° to 2,200° F."

GARDNER F. WILLIAMS, Washington, D. C. (communication to the Secretary, received January 31, 1905):—I beg to call attention to two errors of statement in the section of the Proceedings of the Lake Superior Meeting of the Institute, September, 1904, entitled "General Information of the Geology and Mines of the Lake Superior Copper-District," by Horace J. Stevens, which was republished from the pamphlet prepared and distributed by the Houghton Local Committee.

On page 213, under the heading "*Calumet & Hecla*," is the statement "the mine was opened in 1866 and has paid dividends to date of $85,850,000, paying, at present, quarterly dividends of $1,000,000 each. *No other mining corporation in the world has paid an equal amount in profits.*" This last remark, which I have italicized, is not in accordance with the facts. The De Beers Consolidated Mines, Ltd. (of which I have the honor of being General Manager), was formed March 12, 1888, and from that time until June 30, 1904, it has paid dividends of £20,804,563 (approximately equivalent to $101,000,000). In addition, the Life Governors have received £858,020 out of the

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profits; investments exceeding £1,000,000 have been made in Consols for a reserve fund, and the entire mining-equipment and several mining-properties have been paid for out of the profits. From June 30, 1895, to June 30, 1904, dividends exceeding $71,000,000 have been paid; this period including the time of the Boer war, which materially interfered with the mining operations. A detailed account of these dividends, given in my book, *The Diamond Mines of South Africa*, Appendix V., p. 671, is as follows:—

For year ending June 30, 1902, dividends paid, £2,445,000.
For year ending June 30, 1903, dividends paid, £2,175,000.
For year ending June 30, 1904, dividends paid, £2,175,000.

The second error is on page 215, lines 18 and 19, in the statement "At the Red Jacket shaft the Kimberly system of swinging-skips under cages is in use." At Kimberly, we use the self-dumping skips, which run in balance, but each skip has its own compartment and is not swung under the cage. The cages used for lowering and hoisting the men and for sending supplies into the mines are in compartments of the shafts by themselves. A full description of the hoisting-equipment of the Kimberly mines is given in my book, *The Diamond Mines of South Africa*, p. 323.

Another error appears on page 243 in the Biographical Notices of 1903, in which it is said that the late Victor M. Clement, "Together with Mr. John Hays Hammond, and others, was sentenced to death by the Boers, after the 'Jameson raid,' and purchased his life and liberty with a large sum of money." As a matter of fact, the four leaders only were sentenced to death, namely:—Mr. Lionel Phillips; Mr. (now Sir) George Farrar; Mr. John Hays Hammond and Col. Frank W. Rhodes; the death sentences being later commuted and the men liberated on the payment of $125,000 each. Mr. Clement was sentenced to imprisonment for, I think, two years, and he was released on the payment of a fine of $10,000.

Please do not think me a captious critic, but I feel that you are always desirous of having errors pointed out in any of the publications of the Institute.

Page 236, seventh line from top, "Sheridan Smillie" should be "Sheldon Smillie."

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Features of the Occurrence of Ore at Red Mountain, Ouray County, Colo.

BY T. E. SCHWARZ, DENVER, COLO.

(Annual Meeting, February, 1905.)

THE publication of the report by Mr. F. L. Ransome[1] was welcomed by many engineers who had mined in the heart of the San Juan country, braved its long and snowy winters, climbed its lofty peaks, run the gauntlet of its snow-slides, and studied its problems of geology, while making dividends for their English or American clients. This report helps one to understand why the bonanzas occurred, and why the inevitable borrasca presents itself where least expected.

The Red Mountain district of Ouray county, a very productive section from 1884 to 1893, occupies considerable space in the report mentioned. The character of the ore-occurrence, being distinctive and presenting many features not existing elsewhere, has been deserving of special study. It is to be regretted that Mr. Ransome was unable to observe the conditions underground, as all the principal properties, many of which I opened and developed, were closed at the time of his investigation. It has been my privilege to have contributed a few brief papers,[2] recording my observations in developing these properties; and it is the object of the present paper to present certain sections of the Yankee Girl mine, which throw additional light on the geology of the section and make the available data more complete. The main Yankee Girl chimney or "stock" was the most productive for the first 500 ft. in depth of any chimney in the district.

Mr. Ransome describes these ore-bodies as upright stocks, referring to them as follows:[3]

[1] *A Report on the Economic Geology of the Silverton Quadrangle, Colorado, Bulletin* No. 182, by F. L. Ransome, U. S. Geological Survey.

[2] The Ore-Deposits of Red Mountain, Ouray County, Colo., *Trans.*, xviii., 139–145 (1889–90). Also xxvi., 1056–1059 (1896).

[3] *Op. cit.*, p. 107.

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"It is evident from the descriptions given of the Red Mountain stocks, that the term 'chimney' applied to them is somewhat misleading. They are not simple vertical pipes of ore extending from the surface to indefinite depths, but are separate bodies of irregular, lenticular, or spindle-shaped form, often completely enclosed by country-rock, but linked with neighboring stocks by fissures which are often small and which carry little or no ore."

I cannot agree with Mr. Ransome in the use of the term "stock;" and I regard "chimney" as much more expressive and suggestive of the main features of the occurrence. It seems to me the former term applies no better to the Red Mountain ore-bodies than to those of the Bassick or Bull-Domingo mines of Custer county. One of the features of the ore-bodies of Red Mountain is this very persistence in depth, proven in the Yankee Girl, Congress, Genesee-Vanderbilt, Silver Bell, and Guston mines, to the depths attained. I do not recall a single instance of a chimney on the above-named properties where the terms "lenticular" or "spindle-shaped" would apply as descriptive of the form of the ore-bodies. Their formation by replacement of the breccia is well proven, and the merging of the ore, which occurs massive and quite free from silica or waste, into the country-rock on nearly all sides is characteristic. It is true they were sometimes very erratic and difficult to follow, especially in the upper levels, owing to the absence of any defined slip or fracture leading to the ore-body, or connecting two or more chimneys.

In opening a new level, the "ore-break," or belt of altered breccia in which the chimneys occurred,[4] was first located. In this area there might be some one fracture-plane, perhaps very obscure, which if followed would lead to a chimney; but as such planes rarely had the same course on two successive levels, the finding of the "quartz" or siliceous envelope of the shoot was, in the upper portions of the ore-bodies, the first notice of their proximity. The difficulty of following the "pay" was increased by the utter uncertainty of the position of the areas of massive andesite. (See Fig. 1.) These well-defined masses enclosed no ore-bodies, and were very undulatory, both vertically and horizontally. Whether any of these should be properly described as intrusive dikes, I have never fully satisfied myself, although they possess nearly all the char-

[4] *Trans.*, xxvi., 1056–1059 (1896).

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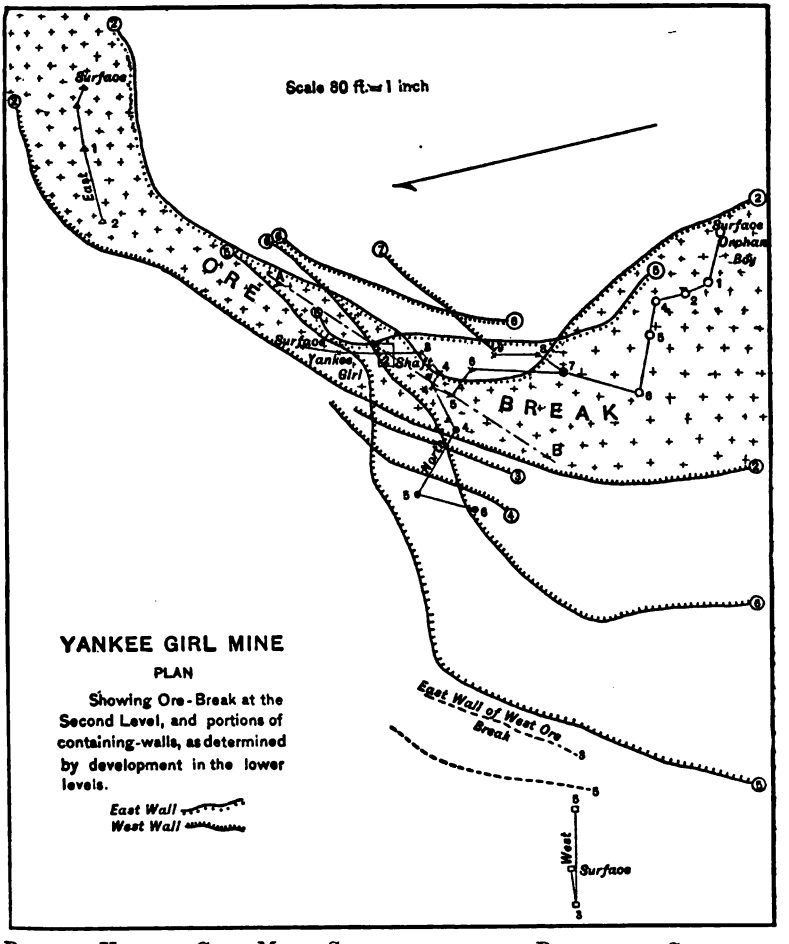
acteristics of a dike. Their course on the surface is more nearly east and west than the course of the general fracturing, which is placed by Mr. Ransome at N. 20° E. This was well illustrated by the Guston chimney, which in its course in the

FIG. 1.



PLAN OF YANKEE GIRL MINE, SHOWING ALSO THE POSITION OF CENTERS OF CHIMNEYS.

Guston-Robinson "ore-break" follows a fracture between the third and fourth levels directly through one of these dike-like masses which extends up the mountain from the Guston into the Little Annie claim. Except in the above-mentioned instance I do not know of any ore-body crossing or entering rock of this character.

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While Mr. Ransome refers to the occurrence of these areas of massive andesite and to intrusive dikes in this section, their relation to each other and to the ore-bodies is not mentioned. I presume, however, these features will be covered by the forthcoming report by Mr. Whitman Cross upon the geology of this district.

One of the distinctive features of these chimneys is the silicified andesite envelope, which, as in the National Belle, Congress, Guston, Robinson, and White Cloud mines, projects in massive knolls of "quartz" above the surrounding surface, and marks their outcrop. This siliceous matrix of the ore-bodies is largest near the surface, decreasing in size and hardness with depth, until it finally ceases to be an envelope of the chimney; and the enclosing rock becomes comparatively soft, altered andesite. Its occurrence seems to be parallel with that of the higher-grade ores, which would indicate that it was a feature of the zone of secondary enrichment.

The accompanying sketch of the Yankee Girl chimneys, given in Fig. 1, shows the position of the center of each ore-chimney on the different levels covered by the notes, and of the bounding-faces of massive andesite, which enclose the ore-break. The development on the successive levels was not sufficient to show the limits of the ore-break at all points; therefore, the position of the andesite faces is outlined only so far as it has been practically disclosed by drifts. It will be observed that the several chimneys gradually approach the largest, or Yankee Girl chimney, in depth. The Orphan Boy and the Yankee Girl chimneys joined between the sixth and seventh levels, as shown in Figs. 1 and 3. On the sixth level, just above the nearly horizontal clay-seam, which is referred to by Mr. Ransome,[5] a distinctly solid body of ore occurred connecting these two chimneys for nearly 6 ft. in height and 60 ft. in length. There was no faulting on this flat seam, as has been stated, but there was a very marked change in the character and grade of the ore. After the Yankee Girl ore-body passed below it, the stromeyerite, which occurs above, ceased entirely, and massive bornite occurs, which is rare above this point.

Referring again to this cross-fracture, I see no reason for

[5] *Op. cit.*, p. 218.

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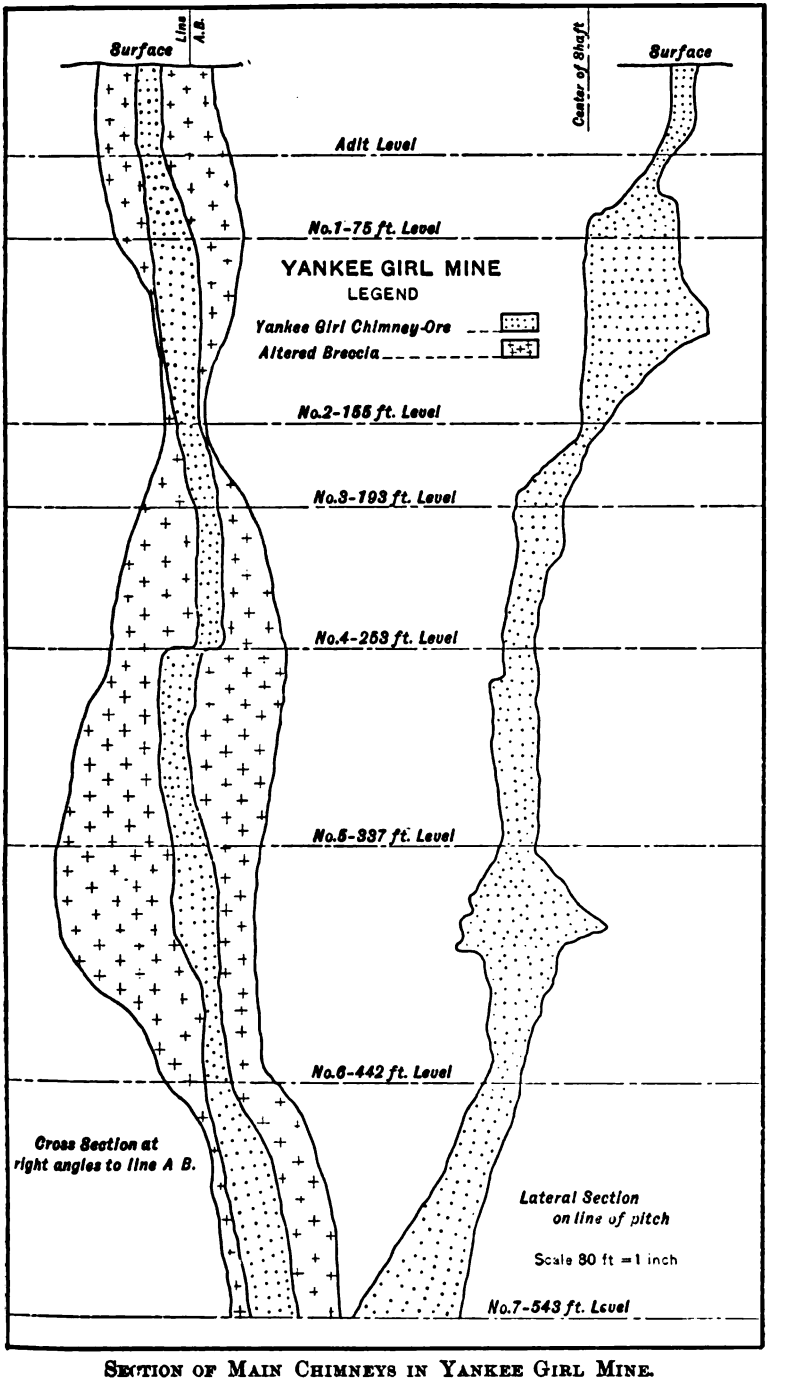
FIG. 2.



SECTION OF MAIN CHIMNEYS IN YANKEE GIRL MINE.

[5]

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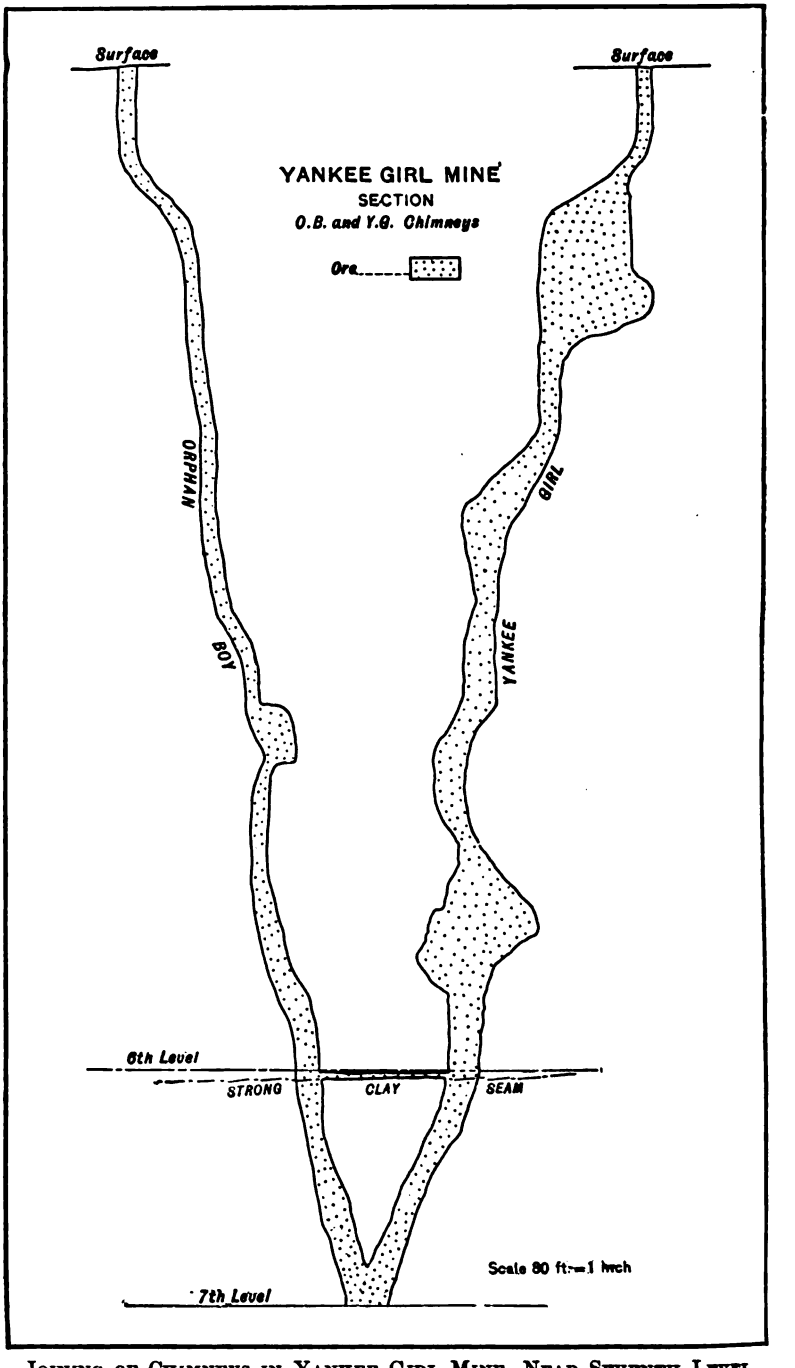
FIG. 3.



JOINING OF CHIMNEYS IN YANKEE GIRL MINE, NEAR SEVENTH LEVEL.

[6]

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assuming that it is identical with the fault-plane which occurs below the seventh level of the Guston mine, as stated by Mr. Ransome.[6] The evidence was that it antedated the ore-deposition, and in fact there are but slight indications anywhere in the mine of displacements or fracturing after the ore was deposited.

It is interesting to note the increasing percentage of bismuth in the ores of the upper levels (from 300 ft. to 500 ft. in depth) of the mines on the line of the main fracturing, from those on the south and highest in elevation to those on the north and on lower ground. In the Congress mine on the extreme south and near the divide, the ores are almost entirely enargite. Proceeding down the valley the Hudson and St. Lawrence ores show appreciable quantities of bismuth, which increase in the Genesee-Vanderbilt, Guston, and Silver Bell mines, in the order named. The latter property produced massive bodies of ore high in bismuth, and carrying from 500 to 1,000 oz. of silver per ton. The ores of the Yankee Girl mine are not included in this comparison, being to the west of the main fracturing mentioned; however, they carried little of the bismuth minerals.

Fig. 2 gives the sections of the main chimney of the Yankee Girl mine from the surface to the seventh level, and shows the ore-body itself, the altered andesite and siliceous envelope filling the ore-break, and the bounding-faces of massive andesite between which the chimneys occur.

There are therefore many features of the ore-occurrence in this section, which should be more fully studied and which might be discussed with profit.

The Yankee Girl chimney is, as Mr. Ransome points out, a notable instance of secondary enrichment. On the theory of descending acid-solutions, the seam referred to as occurring just below the sixth level, evidently induced a concentration of silver-values above that point, at the same time allowing the copper-values to pass through it and enrich the ore-body below, and accounted for an entire change in the mineralogical character of the ore.

What were the ores of the upper portions of the Yankee Girl

[6] *Op. cit.*, p. 108.

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and Guston chimneys, previous to the scoring away to the present level? They showed low-grade lead-ores at surface, followed by increasing percentages of copper, the disappearance of lead, and the marked occurrence of the rich silver, copper and bismuth combinations. Is it not probable that enargite existed in higher horizons, as it forms the ore of the chimneys on higher ground to the south? In the National Belle mine, it occurs as a principal mineral in an ore-body 75 by 60 ft. in area, and appears in all the workings as the sulphides occur in depth. In the Congress mine it is the ore to the lowest level (450 ft.), and it occurs also in the upper workings of the Genesee-Vanderbilt mine.

The paper by Mr. Walter Harvey Weed,[7] explaining the occurrence of enargite in the Butte mines, as a secondary product deposited by ascending alkaline-solutions, is very interesting in this connection, and many of the features of the Red Mountain chimneys seem in accordance with this theory.

I desire therefore to call special attention to the following features of the occurrence of ore in the Red Mountain district, viz.:

1. That the ores have occurred in persistent chimneys, to which the term "stocks" does not well apply.

2. That in the upper zones—from 300 to 500 ft.—well defined fracture-planes, bounding an ore-body or connecting two or more of them, were not much in evidence. The wet fissures filled with clay described by Ransome[8] became marked features only with depth.

3. The masses of blue, blocky, unaltered andesite, resembling dikes, outcrop on the surface with approximately east and west courses, and make an angle from 70 to 90° with the course of the general fracturing.

4. The siliceous envelope of the chimneys is a marked feature on the surface and in the upper levels, gradually disappearing with depth.

5. Enargite occurs but slightly in the Yankee Girl and Guston chimneys, while it is the ore of the chimneys to the south on higher ground.

[7] Ore-Deposition and Vein-Enrichment by Ascending Hot Waters, *Trans.*, xxxiii., 747.

[8] *Op. cit.*, p. 103.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Occurrence of Stibnite at Steamboat Springs, Nevada.

BY WALDEMAR LINDGREN, WASHINGTON, D. C.*

(Annual Meeting, February, 1905.)

THE important investigations of Dr. G. F. Becker at Steamboat Springs, Nev., in 1885, aided by the analytical work of W. H. Melville, established the fact that sulphides were being deposited at the surface by hot ascending waters.[1] Steamboat Springs is situated near the eastern base of the escarpment of the Sierra Nevada, six miles distant from the Comstock lode. In a region of former volcanic activity, hot springs with a temperature of about 80° C. break forth through a fissure in granodiorite, and at the base of a basaltic bluff the waters have deposited a large amount of siliceous and calcareous sinter, some of which is colored red by antimony sulphide. The sinter from one of the springs, analyzed by Mr. Melville,[2] was found to contain the following quantities of the heavy metals:

	Grams per Kilogram.
Ferric oxide,	1.0500
Antimonious and arsenious sulphides,	23.0000
Mercuric sulphide,	0.0020
Cupric sulphide,	0.0130
Lead,	0.0200
Gold,	0.0010
Silver,	0.0004

The relative quantities of antimonious and arsenious sulphides are not given in the above analysis, but I believe that the former greatly predominated.

The water of one of the springs was also analyzed by Melville and the re-calculation of this analysis to salts that were possibly present is given in Table I.

* Published with the permission of the Director of the U. S. Geological Survey.

[1] *Monograph XIII., U. S. Geological Survey*, pp. 331–353.

[2] *Op. cit.*, p. 44.

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TABLE I.—*Possible Composition of the Water of Steamboat Springs, Prior to Oxidation.*[3]

	Grams per 10 Liters.
Ferrous carbonate, $FeCO_3$,	0.0029
Magnesium carbonate, $MgCO_3$,	0.0099
Calcic carbonate, $CaCO_3$,	0.1577
Calcic phosphate, $Ca_3P_2O_8$,	0.0137
Potassic chloride, KCl,	1.9735
Lithic sulphate, Li_2SO_4,	0.5650
Sodic chloride, NaCl,	14.1475
Sodic sulphydrate, NaHS,	0.0358
Sodic sulphate, Na_2SO_4,	1.1147
Sodic bicarbonate, $NaHCO_3$,	2.9023
Sodic monocarbonate, Na_2CO_3,	0.4314
Sodic biborate, $Na_2B_4O_7$,	3.1368
Sodic quadrisilicate, $Na_2Si_4O_9$,	3.9090
Sodic sulphantimonide, Na_3SbS_3,	0.0100
Sodic sulpharsenide, Na_3AsS_3,	0.0866
Alumina, Al_2O_3,	0.0025
Sodium-mercury sulphide, HgS, nNa_2S,	trace

The recalculation given in Table I. is, of course, only of very approximate value, since it is not known with certainty in what condition the different acids and bases are present. Moreover, the salts are partly dissociated. Attention should be called to the fact that the water contains far more arsenic than antimony, but during its filtration a red precipitate formed, consisting of arsenic and antimony sulphides,—in what relative amounts the report does not state.

Having assisted Dr. Becker in his examinations in 1885, I naturally felt a strong interest in this locality, and in 1901 I revisited the place for a few hours, making a few observations which are here recorded.

The flow of water from the springs was found to be greatly reduced, probably on account of clogging of the channel. Several bore-holes had been sunk in order to obtain a better flow, and from these holes fragments of quartz had been brought up which were said to assay high in gold and silver. Of this I know nothing except from hearsay, but at any rate these results seem to have stimulated prospecting activity, for a shaft had recently been sunk to a depth of 30 ft. near the railway-station, on the sinter-flat a few hundred feet away from, and about 20

[3] *Op. cit.*, p. 347.

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ft. above, Steamboat creek, which is the main drainage-line of the valley. After sinking 25 ft. through sinter, a loose sandy gravel was struck containing well-washed pebbles of granite and andesite, which carried so great an abundance of hot water as to lead to the suspension of mining operations. The gravel was said to contain small quantities of gold and silver, but I am not prepared to discuss this aspect of the case; moreover, the assays or the material would have no great value regarding the derivation of the gold and silver if found, for they might have been introduced in different ways. A sample of the same or similar material transmitted to me in the summer of 1904 by Mr. W. H. Weed was assayed and yielded a trace of gold.

Upon examining the dump I found that the gravel throughout contained small shining prisms and particles of metallic luster. A generous sample of the material was collected, but further work was delayed, until 1903, when an examination of the sandy part of the gravel showed it to contain a considerable quantity of stibnite in the form of loose slender prisms of the usual type, up to about 1mm. in length and usually without terminal faces. The prisms are sometimes bent and often combined in radiating groups, and may be observed adhering to the surface of nearly every pebble of the gravel, both large and small. Some of the larger granite-cobbles, which usually are soft and decomposed, contain bunches of stibnite crystals in cracks and crevices. With the exception of clastic magnetite, the only other metallic mineral found in the gravel is pyrite, which forms loose or intergrown crystals of octahedral form, sometimes combined with the cube. In many cases both pyrite and stibnite have crystallized on the surface of pebbles, the former often being tarnished to a black color. Grains of quartz occur with the pyrite, but is not clearly crystallized and may be clastic. A black opaline material containing about 1 per cent. of carbon, according to a determination by Dr. E. T. Allen, sometimes adheres to the andesite pebbles.

In order to obtain an idea of quantitative relationships, the sandy part of the gravel was further examined by Dr. E. T. Allen, of the U. S. Geological Survey, who found that lead, copper, zinc and mercury were absent, and that the material contained antimony, 0.4, arsenic, 0.067, and sulphur, 1.88 per

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cent., which corresponds to stibnite (Sb_2S_3), 0.56; orpiment, (As_2S_3), 0.107; and pyrite (FeS_2), 3.13 per cent.

It is not certain in what form the arsenic is present. Stibnite does not usually contain arsenic and, on the other hand, no separate arsenical mineral could be recognized.

That stibnite and pyrite could be of clastic origin is entirely out of the question, and I believe it absolutely certain that they have been deposited by the hot waters which permeate the gravel. Considering that the waters have been shown to contain a considerable quantity of antimony, the occurrence seems of great interest.

No metallic sulphides corresponding in appearance to the normal minerals were found by Dr. Becker in the sinters, but cinnabar occurs disseminated in the decomposed granite, some distance away from the present springs. The antimonious sulphide, which colors parts of the sinter and always appears red and amorphous, was called metastibnite by Dr. Becker. The absence of the ordinary minerals of metallic luster seemed, in a way, a missing link in the chain of evidence proving the deposition of ores from hot ascending waters, and this link is now supplied by the observations recorded above.

Physical conditions differing very slightly from those at the actual surface will evidently produce crystallized minerals of normal habit and form. Many years ago Senarmont succeeded in forming crystals of stibnite from the amorphous sulphide by heating to 250° C. in closed tube with a solution of sodium carbonate.

During the investigation of the quicksilver deposits of the Pacific slope, Messrs. Becker & Melville found that stibnite is easily crystallized from solutions similar to the waters of Steamboat Springs in sealed tubes heated to about 150° C. This renders it probable that a moderate pressure such as exists at a small distance below the surface is sufficient to induce the formation of the crystals described in this paper. Very likely also the presence of organic matter in the gravel is one of the conditions favoring the deposition.

Many important deposits of stibnite occur in sedimentary rocks in a manner which renders very probable a genesis somewhat similar to that of the occurrence here described.

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Acid Open-Hearth Manipulation.*

BY ANDREW McWILLIAM AND WILLIAM H. HATFIELD, SHEFFIELD, ENGLAND.

(Lake Superior Meeting, September, 1904.)

AT the 1902 May meeting of the Iron and Steel Institute, the authors presented a paper on "The Elimination of Silicon in the Acid Open-Hearth," wherein they recorded a few typical examples of certain Siemens charges out of a very large number specially watched for the purposes of that research. The main deductions drawn from those experiments have been accepted in all the intelligent criticism during or after the discussion on the paper. One opinion, however, was rigidly held by all who mentioned the matter, with the single exception, perhaps, of Mr. Lange, and that was the necessity for the attainment and the maintenance of an abnormally high temperature in order that the percentage of silicon in the metallic bath might increase, or that an unusually large proportion of silicon might be retained in the metal. The original decision was arrived at on the results of scores of trials, and has since been the subject of frequent experiment. The authors still retain their opinion that although, naturally, a higher temperature will accelerate the (probable) action between the carbon of the steel and the excess silica of the slag, they then fairly proved, and have now abundantly confirmed the fact, that around the temperatures occurring in Siemens steel-making practice the chemical composition of the slag, particularly with regard to its acidity, is the factor which determines whether the percentage of silicon in the molten steel shall increase or decrease.

In the discussion in 1902 Mr. Saniter called attention to Mr. H. H. Campbell's work, and in the reply the authors also mentioned that of Messrs. Winder and Brunton of 1892, the two

* Presented at the New York meeting of the Iron and Steel Institute, October, 1904; and here published under a mutual agreement between the Councils of the two Institutes. Mr. Andrew McWilliam, Assoc. R.S.M., is lecturer in metallurgy at University College, Sheffield, and Mr. William H. Hatfield, Mappin Medallist, is metallurgist to Messrs. John Crowley & Co., Meadow Hall Iron & Steel Works, Sheffield.

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latter recording the fact of a rise in silicon, and p. 277, sec. x., *g.*, in the former's excellent work on steel making, reads:

"*Conditions Modifying the Character of the Product.*—If the temperature of the metal is very high, the last traces of silicon will not be oxidized, for the affinity of silicon for oxygen is a function of the temperature. . . . Thus the open-hearth cannot rival the converter in producing high silicon metal by non-combustion, but under suitable conditions the amount carried along in the metal may be quite appreciable, and by holding the bath at a very high temperature with a siliceous slag, there will even be a reduction of the silica of the hearth according to the equation $SiO_2 + 2C = Si + 2CO$."

Several heats bearing on this point have been observed, as, for example, one the details of which have been sent recently to the authors, where the carbon of the bath was 2 per cent. at the start, and by adjustment of the slag the silicon was still at 0.4 per cent. when the carbon had fallen to 1.4 per cent.

Despite Mr. Saniter's dictum in the 1902 discussion, although large additions to the bath undoubtedly reduce its temperature, it is not necessarily overheated because no additions are being made to the slag. Consider for a moment the case of turning off the gas and air. By these means it is quite easy to keep the temperature normal without the charging of cold material into the furnace.

The authors then tried the experiment of allowing the metal and slag to interact with only small additions of ore, the temperature ranging from normal to slightly hot, and samples ran as follows: 9 a.m., carbon, 1.55 per cent.; 9.25 a.m., carbon, 1.35; 9.50 a.m., carbon, 1.20, with manganese 0.09 per cent., and silicon 0.60 per cent.

Still further to test the point in as Faraday-like a manner as possible, it was decided to look around for the most siliceous material lying about, that might reasonably be expected easily to enter the slag, and the choice fell on a heap of old red bricks of the following average chemical composition:—

	Per Cent.		Per Cent.		Per Cent.
Silica,	78.9	Ferric oxide,	4.7	Alumina,	13.3
Lime,	0.4	Magnesia,	1.1		

Eight cwts. of these were thrown into the furnace in three minutes, and after this (to a Siemens charge) "iced drink" the silicon steadily, though slowly, rose in accordance with the subsequently ascertained composition of the slag.

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Every one must admit that this addition would cool the bath, and as a matter of fact it became comparatively cold. The details of the experiment are given in Table I., and it will be noticed that before the special addition the silicon was fairly low and slowly falling, while after this addition, which raised the percentage of silica in the slag and cooled the bath, the silicon percentage gradually increased.

TABLE I.—*Red Brick Charge.*

Time		Composition of Bath.			Additions.		Consistency of Slag.	Composition of Slag.						
Hr.	Min.	C.C.	Si.	Mn.	Materials.	Cwts.		SiO_2.	FeO.	Fe_2O_3.	Al_2O_3.	MnO.	CaO.	MgO.
5	10	.....	.....	.....	Ore.	¾	.....	.....	.....	.....	.....	.....	.....	.....
5	20	.....	.....	.....	Ore.	1	.....	.....	.....	.....	.....	.....	.....	.....
5	35	.....	.....	.....	Ore.	1	.....	.....	.....	.....	.....	.....	.....	.....
5	40	0.45	0.087	.....	.....	.....	Fairly thin.	51.49	24.90	1.74	2.08	17.57	1.10	0.78
5	45	.....	.....	.....	Ore.	1	.....	.....	.....	.....	.....	.....	.....	.....
5	55	.....	.....	.....	Ore.	1	.....	.....	.....	.....	.....	.....	.....	.....
6	0	0.31	0.034	0.039	.....	.....	Moderately thin.	.....	.....	.....	.....	.....	.....	.....
6	2-5	.....	.....	.....	Red brick.	8	.....	.....	.....	.....	.....	.....	.....	.....
6	15	0.22	0.034	0.037	.....	.....	Thickening.	.....	.....	.....	.....	.....	.....	.....
6	25	0.20	0.049	0.039	.....	.....	Thickening.	.....	.....	.....	.....	.....	.....	.....
6	35	0.17	0.049	0.039	.....	.....	Thick.	55.64	21.38	1.27	4.10	15.19	1.38	0.60
6	40	.....	.....	.....	Ore. / Lime.	½ / 1	.....	.....	.....	.....	.....	.....	.....	.....
6	50	0.14	0.053	0.039	.....	.....	Thinning.	.....	.....	.....	.....	.....	.....	.....
7	0	0.18	0.046	0.04	.....	.....	Thinner.	53.98	21.74	0.86	4.43	14.92	2.10	2.08

Owing to an apparently abnormal behavior of the slag in certain trials where a considerable quantity of lime had been added, a special charge was run with the object of watching more closely the effect of lime in the slag on the silicon-content of the bath. The slag in one heat was allowed to thicken from 5.50 p.m. to 6.40 p.m., when at 6.41 p.m. 3.5 cwts. of limestone were added. This addition had the effect of thinning (increasing the fluidity of) the slag and making it appear more basic than it really was, comparing its fluidity with that of similar slags without lime. Also, with lime present, the balance-point seems to be slightly altered, perhaps owing to the lime not being an oxidizing base, and with 53.5 per cent. silica in the slag, the silicon is not lowered but is slowly increasing. This with other results from the experiment, bearing out the general trend of trials made with other ends in view, and hence not followed out in detail at the time, was useful in the making of certain special steels for commercial purposes, but should also be

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of general interest as an aid to the study of the reactions between metal and slag. The necessary details are given in Table II.

TABLE. II.—*Lime Charge.*

Time		Composition of Bath.			Additions.		Consistency of Slag.	Composition of Slag.						
Hr.	Min.	C.C.	Si.	Mn.	Materials.	Cwts.		SiO_2.	FeO.	Fe_2O_3.	Al_2O_3.	MnO.	CaO.	MgO.
5	50	0.54	0.025	0.049	.....	.....	Fairly thin.	.....	.....	.....	.....	.....	.....	.....
6	0	0.53	0.031	0.055	.....	.....	Thickening.	.....	.....	.....	.....	.....	.....	.....
6	20	0.44	0.040	0.064	.....	.....	Thickish.	.....	.....	.....	.....	.....	.....	.....
6	40	0.36	0.049	0.074	.....	.....	Thick.	55.45	18.49	1.61	2.02	20.77	2.09	0.89
6	41	.....	.....	.....	Lime.	3½	.....	.....	.....	.....	.....	.....	.....	.....
6	50	0.30	0.051	0.081	.....	.....	Thinning.	.....	.....	.....	.....	.....	.....	.....
7	0	0.27	0.052	0.096	.....	.....	Thin.	53.54	16.59	1.36	1.61	19.77	5.96	1.05

TABLE III.—*Magnesia Charge.*

Time.		Composition of Bath.			Additions.		Consistency of Slag.	Composition of Slag.	
Hr.	Min.	C.C.	Si.	Mn.	Materials.	Cwts.		SiO_2.	MgO.
3	15	0.44	0.026	0.045	.....	.....	Thinnish.	.....	.....
3	30	0.40	0.029	0.057	.....	.....	Thickening.	.....	.....
3	45	0.29	0.034	0.063	.....	.....	Fairly thick.	55.72	3.0
3	50	.....	.....	.....	Magnesia.	3	.....	.....	.....
4	10	0.19	0.035	0.067	.....	.....	Thinning.	.....	.....
4	28	0.17	0.037	0.074	.....	.....	Thin.	54.03	4.0

At this stage it seemed "as if increase of appetite had grown with what it fed on," for these results created the desire to know a little of the effect of other and more unusual bases on the slag and steel, so another special heat was run, using magnesia instead of lime.

It will be seen that the magnesia acts after the same fashion as the lime, with the special characteristics perhaps not quite so well defined.

To throw some light on the rather different effect produced by these basic oxides which do not so readily give up their oxygen compared with the oxides of iron, a charge in which the slag should be allowed to thicken as before and then be thinned with the very oxidizing manganese dioxide was determined on.

In Table IV. are given those details of this charge which are of interest in connection with the experiment. No iron-ore was

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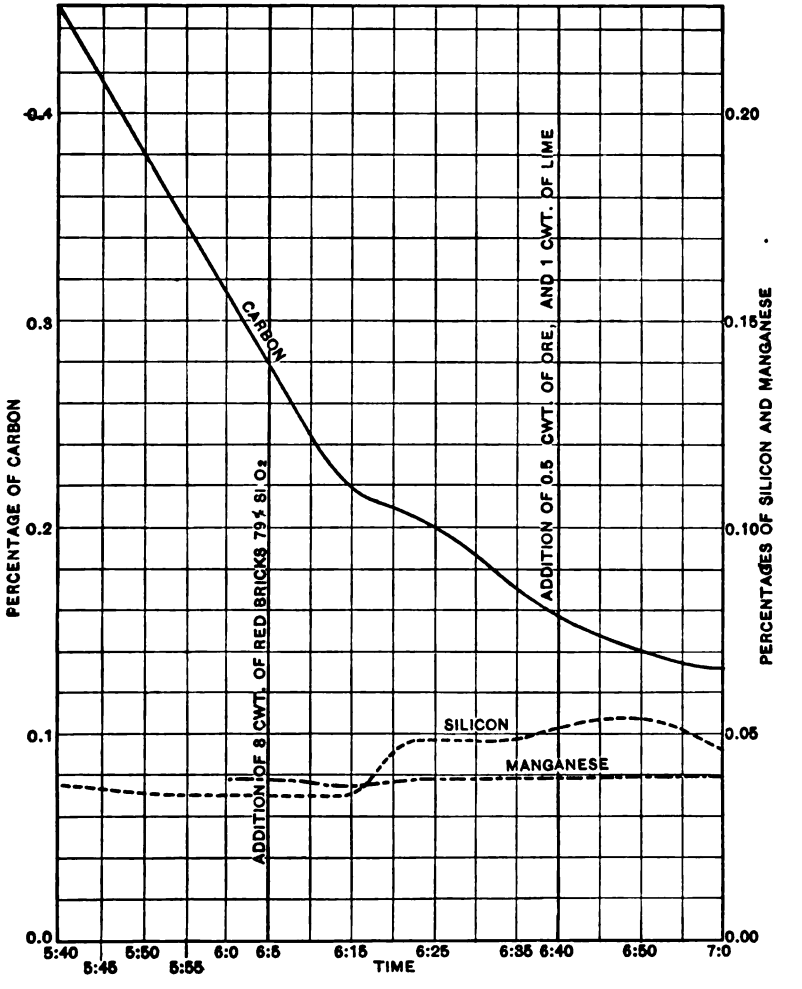
added within the limit of the table, the charge was tapped at 3:10 and behaved normally. It will be seen from the analyses that the carbon continued to fall satisfactorily the whole time, that the 520 lb. of the black oxide of manganese thinned the

FIG. 1.



THE RED-BRICK CHARGE.

slag and made it less siliceous for a short time, but that the new slag quickly helped itself to silica and became viscous and highly siliceous. The source of that silica was evident after tapping, for the appearance of the bottom suggested that as a

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process for the production of small-scale models of the Grand Cañon of Colorado, inaccurate in detail, but faithful to the spirit, it might be recommended, but as a means of making even special steels it would be a little too exciting. From this it will readily be understood why the effect of the more basic slags of the last two series have not been tried and are not likely to be, at least by the present experimenters.

TABLE IV.—*Pyrolusite, 22-Ton Charge.*

Time.		Composition of Bath.			Additions.		Consistency of Slag.	Composition of Slag.			
Hr.	Min.	C.C.	Si.	Mn.	Materials.	Lb.		SiO_2.	MnO.	CaO.	MgO.
11	45	.....	0.20	.....	....	.....	Thin.	.....	.....	.....	.....
12	30	0.74	0.05	0.104	.....	.....	Thickening.	.....	.....	.....	.....
1	0	0.51	0.051	0.097	....	.....	Thickening.	56.74	16.45	2.96	trace
1	15	.....	0.063	.....	....	.....	Thickening.	.....	.....	.....	.....
1	30	0.42	0.069	0.09	.....	.....	Thicker.	.....	.....	.....	.....
1	45	0.34	0.079	0.111	.....	.....	Very thick.	57.62	17.48	.....	.....
1	48 to 2 0	.....	.....	.....	MnO_2.	440	.....	.....	.....	.....	.....
2	5	0.29	0.061	0.115	.....	.....	Thinning.	56.7	18.14	.....	.....
2	10	.....	.....	.....	MnO_2.	80	.....	.....	.....	.....	.....
2	20	0.25	0.052	0.109	.....	.....	Thinning.	55.8	17.4	.....	.....
2	35	0.19	0.066	0.124	.....	.....	Thin.	57.2	18.4	.....	.....
2	50	.....	0.071	.....	.....	.....	Thickening.	.....	.....	.....	.....
3	5	.....	(0.09)	.....	.....	.....	Thickening.	64.06	17.97	.....	.....

An interesting discovery which is at present engaging the attention of the authors was made during a study of the general notes made of any probable points of interest belonging to the special heats, although these points might not seem to have reference to the more immediate question under observation. It will be remembered that at the May meeting of this Institute Mr. Wahlberg read a paper on some of Mr. Brinell's work on "The Influence of Chemical Composition on Soundness of Steel Ingots," in which Brinell showed that in his practice giving certain relative values for silicon and aluminium in terms of manganese, calculating all out to manganese and taking the total, he obtained a number which he named the density quotient, of such a nature that for the same density quotient he obtained the same type of ingot; for a lower quotient one with more blow-holes, and for a higher quotient, fewer. Now the general experience of the authors has corroborated that of Brinell, but during the progress of this research they made the interesting discovery that those charges treated to one hour's

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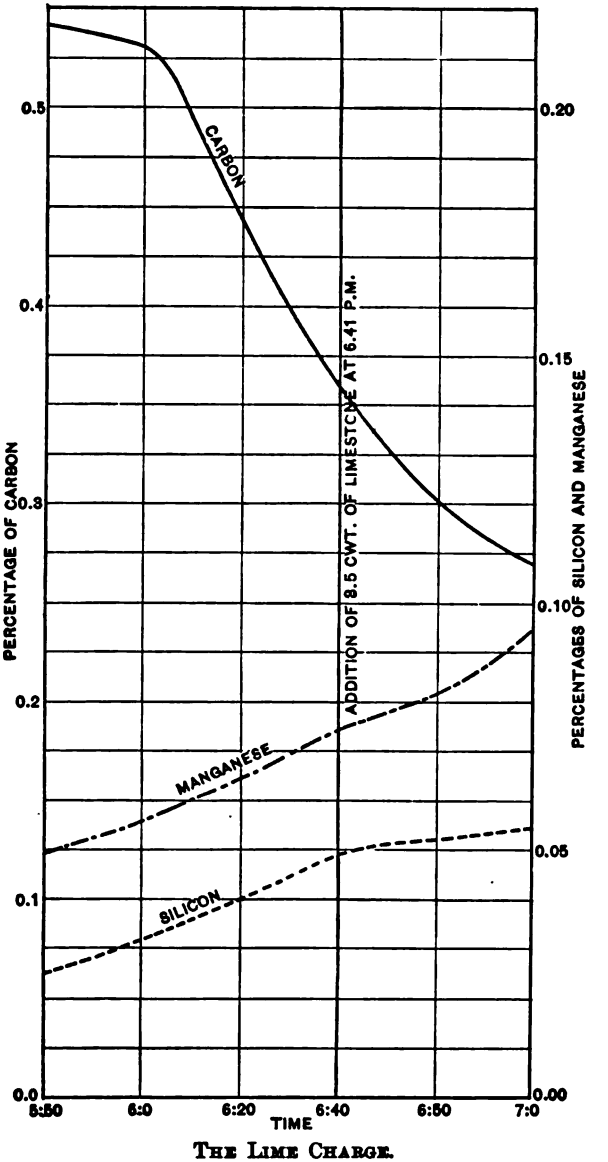
FIG. 2.



THE LIME CHARGE.

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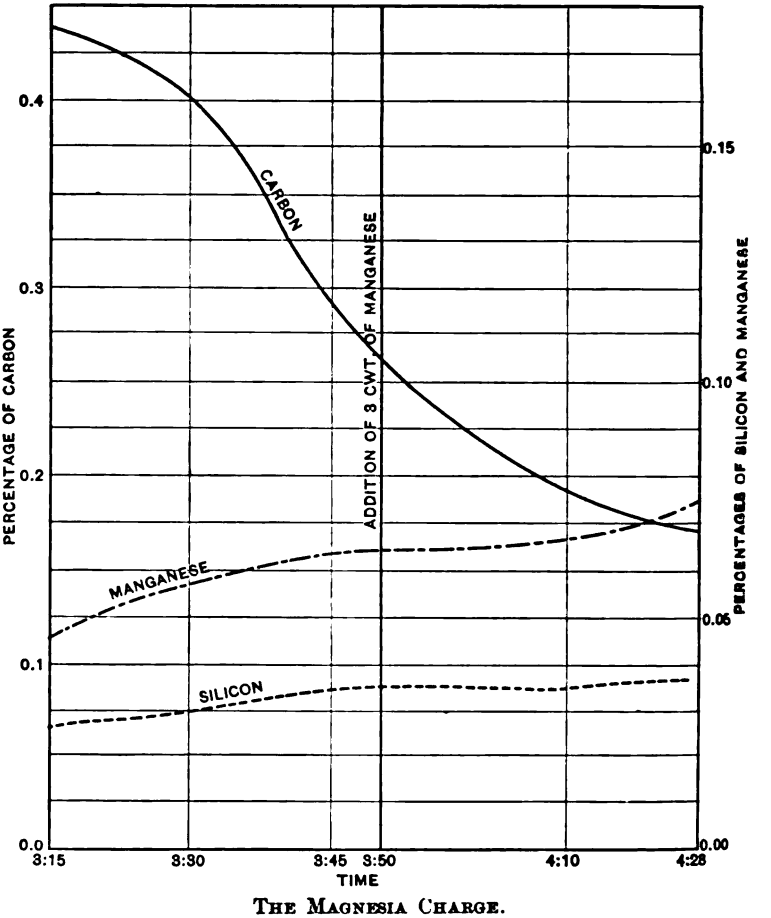
thickening of the slag required a lower density quotient than that for normal heats, to give a certain type of ingot; charges treated to two hours' thickening required a lower number still, while in a heat run specially fast and finished with a much

FIG. 3.



THE MAGNESIA CHARGE.

higher density quotient the ingots corresponded to a lower number.

The following figures, and indeed the whole point raised, are given tentatively as a matter of interest, for it appeals to the writers as being of considerable importance, and they are engaged in following the matter up through this and other channels.

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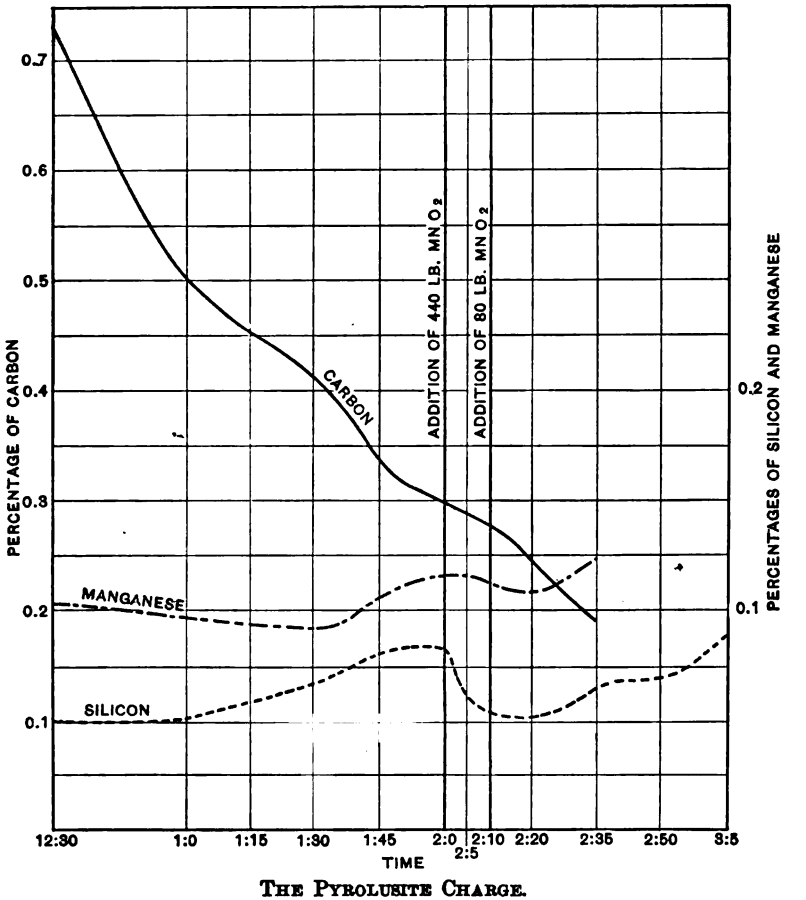
Treatment of Charge.	Density Quotient.	Type of Ingots.
Thickening, 2½ hours, . . .	0.814	good
Thickening, 1½ hours, . . .	1.00	good
Ordinary, ...	1.35	good

The authors have a theory as to the cause of these differences, but prefer to seek further experimental support before making it public.

FIG. 4.



THE PYROLUSITE CHARGE.

Again the authors tender their best thanks to willing helpers; to Messrs. J. H. G. Monypenny, H. H. Slater, M. H. Graham and J. Kilby, for their very careful analytical work, and to Messrs. John Crowley & Co., Ltd., Meadow Hall Iron and Steel Works, Sheffield, England, for the use of a 25-ton furnace, for the purposes of this research.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Comparison of Methods for the Determination of Carbon and Phosphorus in Steel.

BY BARON JÜPTNER VON JONSTORFF (AUSTRIA), ANDREW A. BLAIR (UNITED STATES), GUNNAR DILLNER (SWEDEN), AND J. E. STEAD, F.R.S., MEMBER OF COUNCIL (ENGLAND).*

(Lake Superior Meeting, September, 1904.)

INTRODUCTION.

IT is a well-known fact that the results of different analysts, when operating on the same identical sample of steel or iron, are far from concordant, and it not infrequently happens that great annoyance and trouble in the commercial community are occasioned by these discrepancies.

Attempts have been made at various times to investigate the causes of such differences and to obtain more reliable methods of analysis. The most important of these was made by the British Association in 1889. The following gentlemen, with Sir W. Roberts-Austen as Chairman, and Professor T. Turner as Secretary, were constituted as a committee to investigate the question, viz.: Sir F. A. Abel, Mr. Edward Riley, Mr. G. J. Snelus, Mr. John Spiller, Professor J. W. Langley, and Professor Tilden.

Average samples of steel were prepared in America under the superintendence of Professor J. W. Langley. The steels, in the form of turnings, were divided and distributed to subcommittees in Great Britain, America, France, Germany, and Sweden.

In Sweden Professor Åkerman acted as Secretary to the Swedish committee. The Government Department of Tests was appointed in Germany; Mr. Ferd. Gautier in France; the British Association Committee in England; Messrs. William

* Presented at the New York meeting of the Iron and Steel Institute, October, 1904; and here published under a mutual agreement between the Councils of the two Institutes.

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Metcalf, Thomas Rod, A. E. Hunt, and Professors J. W. Langley, A. B. Prescott, and M. E. Cooley in America.

The material was furnished gratuitously by the Crescent Steelworks, Pittsburg. The ingots were made of crucible steel, excepting No. 4 standard, with 0.15 per cent. of carbon, which was of open-hearth material. All the samples were turned down and about 50 lb. of turnings from each sample were obtained.

Nos. 1, 2 and 3 standards were from unworked crucible ingots, and No. 4 standard was from a rolled billet.

Portions of each were sent to the several committees and tested by the appointed analysts. It is to be regretted that neither the German nor French committees sent in any results. The Swedish, American and English committees took an active interest in the work, and analyses were sent in to the British Association Committee, who published no less than six reports. The importance of the work done has unfortunately not been fully recognized, and little is really known of what was done by other people apart from the members of the committees. Members of the Iron and Steel Institute, who are interested in the subject, should remember that samples of the standard steels can still be obtained from Professor Thomas Turner, the University, Birmingham, together with the reports of the various committees.

The analysts appointed in England consisted of the following gentlemen, viz.: Mr. A. H. Allen of Sheffield, Mr. W. Jenkins of Dowlais, Mr. G. S. Packer of Glasgow, Mr. J. Pattinson of Newcastle-on-Tyne, Mr. E. Riley of London, Mr. J. E. Stead of Middlesbrough, and the Royal School of Mines, London.

The mean results of the three committees, already mentioned, are as follows:—

Mean Results of the Analyses by the Swedish Committee.

Standard.	No. 1.	No. 2.	No. 3.	No. 4.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Carbon	1.450	0.840	0.500	0.170
Silicon	0.357	0.185	0.150	0.015
Sulphur	0.008	0.004	0.006	0.048
Phosphorus	0.022	0.015	0.021	0.102
Manganese	0.282	0.145	0.170	0.130

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Mean Results of the Analyses by the American Committee.

Standard.	No. 1.	No. 2.	No. 3.	No. 4.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Carbon	1.44	0.80	0.454	0.10
Silicon	0.270	0.202	0.152	0.015
Sulphur	0.004	0.004	0.004	0.038
Phosphorus	0.016	0.010	0.015	0.088
Manganese	0.254	0.124	0.140	0.098

Mean Results of the Analyses by the British Committee.

Standard.	No. 1.	No. 2.	No. 3.	No. 4.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Carbon	1.414	0.816	0.476	0.151
Silicon	0.263	0.191	0.141	0.008
Sulphur, not more than	0.006	0.007	0.008	0.039
Phosphorus	0.018	0.014	0.021	0.078
Manganese	0.259	0.141	0.145	0.130

Mean Results of All the Samples.

	Swedish.	American.	English.
	Per Cent.	Per Cent.	Per Cent.
Carbon	0.740	0.715	0.714
Silicon	0.150	0.160	0.153
Sulphur	0.0165	0.0145	0.017
Phosphorus	0.040	0.032	0.033
Manganese	0.182	0.154	0.169

The only difference of any moment here shown is in the carbon, which, in the Swedish analyses, is about 0.025 per cent. higher than the American and English.

The higher phosphorus shown by the Swedish analyses is due to a very high phosphorus in No. 4 sample. The other analyses closely agree with those of the English and American analysts. Possibly the higher phosphorus in No. 4 may have been due to variations in the sample, which was obtained from a large open-hearth ingot after rolling to a billet.

At the autumn meeting of the Iron and Steel Institute held in Glasgow in 1901, a paper was read by Mr. Axel Wahlberg of Stockholm, on The Variations of Carbon and Phosphorus in Steel Billets. A number of analytical tests were given which were obtained by testing identical samples of steel by several analysts in Sweden; at the testing laboratory of the Royal

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Technical University of Stockholm, by Mr. Gunnar Dillner, and by experts in the Hammarström Laboratory at Kopparberg, in Vienna by Baron Jüptner von Jonstorff, and in England by Mr. J. E. Stead at Middlesbrough. The report gave a full account of the results obtained by the several analytical authorities. On comparing the results there appeared to be rather a striking lack of uniformity beyond what might have been expected, caused presumably by the imperfections of some of the methods of analysis employed. It was suggested by Mr. Wahlberg that it was desirable that a standard international analytical method should be recognized by the Iron and Steel Institute.

In the course of the subsequent discussion it was then suggested by Mr. J. E. Stead, that on account of the differences in the results, there seemed to be some necessity for further research; and proposed that the same analytical authorities should be constituted as a committee, who should make further investigations, exchange their methods of analysis, and endeavor, if possible, to find out which was the best and most reliable.

The proposal was adopted, and the committee was formed consisting of the proposer of the resolution, together with Baron Jüptner von Jonstorff and Mr. Dillner as members. Subsequently Mr. Andrew A. Blair, representing America, was also appointed a member of the same body.

Programme of the Present Researches.

Conforming to proposals made, homogeneous samples of borings were taken from the lower ends of ingots containing respectively about 0.1 per cent., 0.5 per cent., 1.0 per cent., and 1.5 per cent. of carbon. Samples of each steel were then distributed to the respective analysts. Each member of the committee was asked to test his samples by his own approved method, and also by the methods employed by the other analysts.

To the courtesy of the Fagersta Steelworks the committee is indebted for the material required for the analyses—namely, four square bars of acid open-hearth steel containing about the necessary amount of carbon.

Instead of making a general report, it has been considered desirable that the remarks of each member of the committee

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should be independent, and a review or summary of the results given at the close of the papers.

Method for the Determination of Carbon.

BY BARON H. VON JÜPTNER.

Weigh from 1 to 5 grams according to the percentage of carbon. The turnings are treated with from 150 to 300 cc. of neutral solution of copper-ammonium chloride and are dissolved in the cold with continual shaking. If, when this is in progress, basic salts separate out, hydrochloric acid is added drop by drop. The solution is filtered through asbestos which has previously been ignited and washed free from chlorine. The asbestos is placed with the carbonaceous residue in a test-tube with a cooling-tube inserted, the apparatus is tested to see if it is air-tight, then the air contained in the apparatus is sucked out, whereby the current of air passes through sulphuric acid and soda-lime. Next, 20 cc. of chromic acid solution and 100 cc. of sulphuric acid are introduced into the apparatus and the liquid is boiled until no more carbonic anhydride is given off. Air is then drawn through the apparatus, and the U-tube with soda-lime, and finally the accompanied calcium chloride tube, are weighed. The increase of weight multiplied by 0.2727 equals the carbon.

Remarks.

Copper Ammonium Chloride Solution.—300 g. of copper-ammonium chloride to 1 liter of water.

Chromic Acid Solution.—2 : 1. It is prepared with 10 cc. of sulphuric acid to 1 liter, and boiled.

Sulphuric Acid.—2 : 1. It is saturated with chromic acid and boiled.

The apparatus is illustrated in Fig. 1.

Method for the Determination of Phosphorus.

BY BARON H. VON JÜPTNER.

Two grams of steel are dissolved in 60 cc. of nitric acid of 1.2 specific gravity, oxidized with a sufficient quantity of potassium permanganate ($K_2Mn_2O_8$) (moderately concentrated solution) boiled for at least 10 minutes, the manganese dioxide dissolved

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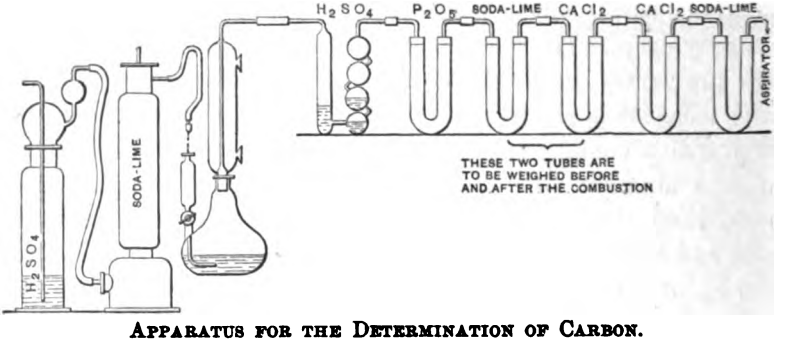
with a little tartaric acid rendered alkaline by ammonia then weakly acidified by nitric acid. After cooling, 50 cc. of molybdenum solution are added, and at a temperature of 50° C. it is allowed to settle (in the water-bath), filtered and well-washed with ammonium nitrate and nitric acid. The precipitate is dissolved off the filter with ammonia, evaporated in a platinum dish (and, if necessary, again filtered), acidified with nitric acid, evaporated to dryness, ignited and weighed. Factor, 0.0163.

FIG. 1.



APPARATUS FOR THE DETERMINATION OF CARBON.

Potassium Permanganate ($K_2Mn_2O_8$).—Fairly concentrated solution.

Tartaric Acid.—Fairly concentrated solution.

Molybdenum Solution.—320 g. of molybdenum acid, 450 cc. of ammonia, 1,050 cc. of water, and 4,800 cc. of nitric acid (1.2 specific gravity).

Washing Liquid.—20 g. of ammonium nitrate, 100 cc. of water, and 10 cc. of concentrated nitric acid.

REPORT BY BARON H. VON JÜPTNER.

Carbon Determinations.

These were carried out in the following manner:—

(*a*) The steel drillings were dissolved in from 50 to 75 cc. of copper sulphate solution (40 g. of $CuSO_4$ in 100 cc. of water). To this was added 100 cc. of concentrated chromic acid and 100 cc. of sulphuric acid (1 : 1), and the carbon was then oxidized. The apparatus employed for this purpose was the ordinary Corleis apparatus.

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The gases were passed (1) through a cooler, (2) through a combustion-tube with platinum quartz (the layers being 15 cm. long)—this latter operation was in accordance with the suggestion of my assistant, Mr. A. Wagner,—(3) through a calcium chloride tube, (4) through a potash apparatus. (5) They were dried near the aspirator with sulphuric acid. The air drawn through was sucked through a system of soda-lime tubes.

(*b*) The arrangement of the apparatus and the procedure are similar to the above, only that instead of the calcium chloride tube (*a*, 3), a Schmitz U-tube and meta-phosphoric acid and sulphuric acid is inserted.

(*c*) The steel drillings were dissolved in 250 cc. of a solution of copper chloride and potassium chloride ($CuCl_2 + KCl$) (160 g. of copper chloride, 90 g. of potassium chloride, and 40 cc. of hydrochloric acid to 1 liter of water), the separated carbon was filtered over asbestos, freed from acid by washing with dilute hydrochloric acid and then with water, dried and burnt in a platinum boat in the combustion-tube in a current of oxygen.

The gases of combustion were passed (1) through a U-tube with copper chloride in the one leg and with potassium bichromate, moistened with a few drops of sulphuric acid, in the other leg; (3) and (4) through two calcium chloride tubes; (5) and (6) through two soda-lime tubes; (7) through a safety tube.

(*d*) The method was the same as that described under (*c*), with the exception that the dissolving was effected in a solution of neutral copper-potassium chloride.

Phosphorus Determinations.

(*a*) The sample, after roasting and dissolving in nitric acid, was dissolved with hydrochloric acid. This was displaced by nitric acid, the silica was filtered off, and the phosphoric acid in the filtrate was precipitated with molybdenum solution, with the addition of ammonium nitrate. After standing for 6 hours in a temperature of from 40° to 50° C. the precipitate was collected upon a weighed filter which had been dried at 110° C., and was washed with water containing ammonium nitrate, slightly acidulated with nitric acid. Finally it was dried at 110° C. and weighed.

(*b*) The method is the same as that described under (*a*), except that the precipitate was dissolved in ammonium hydrate

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evaporated in a weighed crucible, acidulated with a small quantity of nitric acid, dried and weighed.

TABLE I.—*Results Obtained by Baron H. von Jüptner.*

Carbon Determinations.				
Method Employed.	H. 971.	M. 76.	H. 739.	L. 422.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
A (*a*)	1.570	1.105	0.459	0.141
A (*b*)	1.520	1.071	0.439	0.0936
A (*c*)	1.439	1.046	0.424	0.140
A (*d*)	1.450	1.068	0.459	0.142
Phosphorus Determinations.				
B (*a*)	0.032	0.034	0.028	0.028
B (*b*)	0.029	0.030	0.025	0.026

METHOD FOR THE DETERMINATION OF CARBON.

BY ANDREW A. BLAIR.

Place 1 g. of pig-iron, spiegel, or ferromanganese in a 400-cc. Griffin's beaker, and add 100 cc. of a saturated solution of potassium-cupric chloride and 7.5 cc. of hydrochloric acid. For steel or puddled iron, use 3 g. and add 200 cc. of potassium-cupric chloride solution and 15 cc. strong hydrochloric acid. Stir the solution constantly with a glass rod for some minutes at the ordinary temperature. The more it is stirred, the more rapid will be the solution of the iron and of the precipitated copper. The beaker, carefully covered, may now be placed on the top of the air-bath or on a cool part of the sand-bath, but the solution should never be heated hotter than 60° or 70° C., and it should be stirred as often as practicable.

As the most tedious part of the determination of carbon in steel is frequently that which has to do with the decomposition of the steel and the solution of the precipitated copper, particularly in the case of low steels, the samples being nearly always in lumps, and it not being desirable to separate these larger particles for fear that the fine stuff alone may not represent a true average, the machine shown in Fig. 2 is very useful.

It consists of a framework, *A*, of brass, cast in one piece for the sake of rigidity. It is fastened to the table by lugs and

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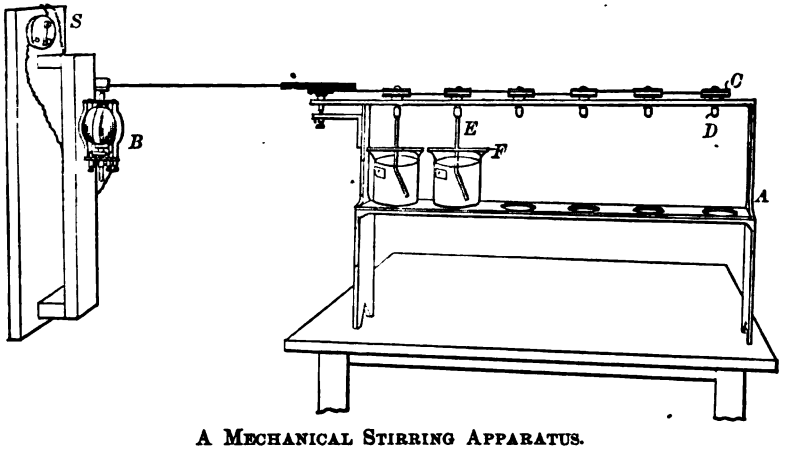
screws not shown in the cut. The shelf on which the beakers stand has on it a piece of asbestos board with holes to fit exactly the bottoms of the beakers to prevent them from moving. To further increase the stability of the beakers (which should be of very heavy glass) their bottoms are ground on a glass plate with fine emery until they have a good bearing surface all around.

The tops, which are covered when on the machine with a plate of glass *F* ground on one side and perforated to allow the passage of the stirring-rods *E*, are likewise ground, so that when slightly moistened the ground glass prevents almost entirely all movement of the covers on the beakers when the machine is in motion.

FIG. 2.



A MECHANICAL STIRRING APPARATUS.

The small wooden pulleys *C* are fitted with brass spindles, which run through the upper cross-piece, and have on their lower ends pieces of rubber tubing, which serve to hold the stirring-rods. The stirring-rods are bent, as shown in the cut, to give the proper motion to the liquid. A small motor *B* adapted to the strength of the current, furnishes the requisite power. The motor, if properly wound, may be attached to an ordinary incandescent lighting current, but a sewing-machine motor run by a dipping battery of three bichromate cells is sufficient to give the necessary number of revolutions.

The fact that it is not only unnecessary to use a neutral solution, but that the use of a neutral solution gives inaccurate

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results, seems now to be thoroughly established by the experiments of the American members of the International Steel Standards Committee. The best practice is to add about 10 per cent. of hydrochloric acid to the solution of the potassium-cupric chloride. The reactions occurring may be considered as

$$Fe + CuCl_2 = FeCl_2 + Cu; \text{ and } Cu + CuCl_2 = 2CuCl.$$

The part taken by the potassium chloride does not seem very clear, but the fact remains that the precipitated copper is much more soluble in the double salt than in any other menstrum. When the precipitated copper is all, or very nearly all, dissolved, which is usually the case in half an hour after the solution of potassium-cupric chloride is added to the drillings, run a little of the acidulated double chloride around the sides of the beaker by means of the rod, wash the rod over the beaker with a jet of water, and let the beaker stand for a few minutes to allow the carbonaceous matter to settle. The best form of filtering-apparatus is shown in the annexed sketches. It consists of the perforated platinum boat, Fig. 3, which fits in the platinum holder. To prepare the boat for use, place it in the holder, as shown in Fig. 4, attach the pump, but do not start it. Fill the boat with prepared asbestos suspended in water, pour enough around the outside of the boat to fill the space *a*, Fig. 4, and start the filter-pump. Continue pouring the suspended asbestos into the space *a*, Fig. 4, until enough is drawn into the joint to make a good packing. By pressing it in all around with a spatula the joint may be made very tight. Pour enough of the suspended asbestos into the boat to make a good, thick felt, and press it down firmly all over the bottom of the boat with something like the square end of a lead pencil, to make it compact. Detach the pump, remove the boat from the holder carefully so as to leave the packing on the sides of the holder, and move it up with the end of a spatula, so that it will remain as shown in Fig. 3. Place another boat in the holder, press the packing into the joint *a*, Fig. 4, with the end of a spatula, fill the boat with suspended asbestos, and start the pump. If necessary, pour a little of the finer suspended asbestos fiber into the joint to make it perfectly tight, and prepare the felt in the boat as before. Dry the boats, and ignite

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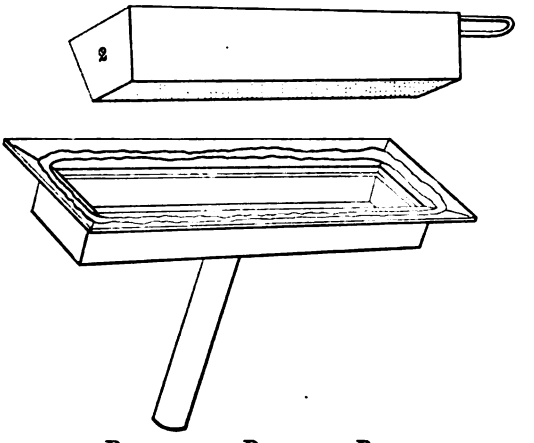
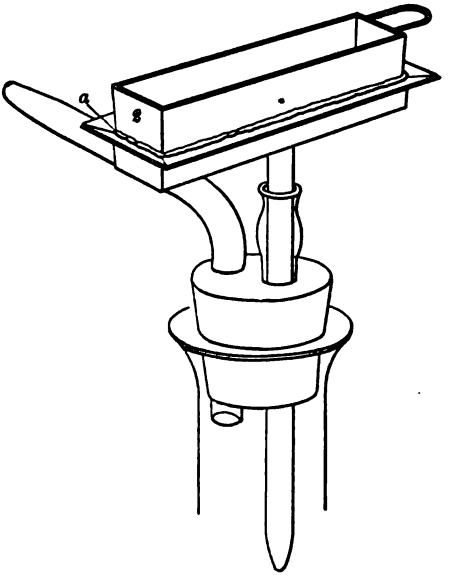
them in the combustion-tube, two at a time, in a current of oxygen. Fit one of these prepared boats in the holder, press

FIG. 3.



PERFORATED PLATINUM BOAT.

FIG. 4.



PERFORATED PLATINUM BOAT ATTACHED TO FLASK.

the packing into the joint as before, first moistening it slightly if it has become dry, start the pump, and pour into the boat

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enough suspended asbestos which has been ignited in oxygen, to form a thin film on the top of the felt. (Barba suggests adding to the solution ignited asbestos in water to make the carbonaceous matter settle and to prevent its clogging the filter. This is a most admirable suggestion and should be generally adopted.) This asbestos film will hold the silica, ferric phosphate, etc., from the carbonaceous residue; and, after the combustion, will usually turn up at the edges, so that it can readily be detached from the main felt, leaving the boat ready for another filtration.

The boat being thus prepared, pour into it the solution of the iron or steel, guiding the stream by a small glass rod held against the tip of the beaker. The solution, if the joint *a*, Fig. 4, is tight, and the pump works well, will usually run through the felt as rapidly as it can be poured into the boat. When the supernatant fluid has all run through, transfer the carbonaceous matter to the boat by a fine stream of cold water from a washing-flask. Pour into the beaker about 10 cc. of dilute hydrochloric acid, run it all around the inside of the beaker by means of the rod to dissolve any adhering salt, wash the glass rod, and wash down the sides of the beaker with a jet of water, and decant the acid into the boat, filling it almost up to the edge. Wash the carbonaceous matter in the boat thoroughly with hot water by filling the boat from the beaker and allowing it to suck through dry, but do not attempt to throw a jet of water into the boat from the washing-flask, as it will be almost certain to throw some of the carbonaceous matter from the boat, or cause it to crawl over the side. In decanting the water from the beaker, the lip must not be allowed to touch the surface of the liquid in the boat, as a film of carbonaceous matter will run up the inside of the beaker. Pour a little dilute acid into the joint between the boat and the holder, allow it to suck through the packing, and wash it several times with hot water. The carbonaceous matter from pig-iron, puddled iron, spiegel, ferromanganese and ingot-steel usually washes like sand, but that from steel which has been hardened, tempered, hammered, or rolled is apt to be more or less gummy, stopping the filter and rendering the filtration and washing prolonged and tedious. It is also apt to adhere more or less to the sides of the beaker, and must be wiped off by a little wad

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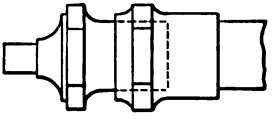
of ignited fibrous asbestos, held in a pair of platinum-pointed forceps, like those shown in Fig. 5.

This wad is then placed in the boat. When the carbonaceous matter is thoroughly washed and sucked dry, detach the pump, remove the boat from the holder, wipe the outside carefully with a piece of silk, place it in a dish covered with a watch-glass, and dry it in a water-bath or in an air-bath at 100° C. When dry, insert the boat in the platinum tube, Fig. 7, and burn off the carbon. The rear end of the tube has a ground joint (see Fig. 6), which may be made perfectly air-tight. The tube has a strengthening band of German silver at *B*, Fig. 7, and the part *P*, which is of phosphor-bronze, is ground in. To prevent the tube from sagging when it is hot, the rear end is supported at *P*, Fig. 7, by a wire from the top of the hood. A piece of platinum gauze 0.5 in. long (12 mm.), rolled up rather loosely, fills the forward end of the tube, then the tube is filled for a distance of about 6 in. (150 mm.) with granular copper

FIG. 5.

PLATINUM-TIPPED FORCEPS.

FIG. 6.



GROUND JOINT.

dioxide, followed by another piece of platinum gauze of the same size as the first, and a similar roll 2 in. long, with a loop, is pushed in after the boat, the rear end coming just forward of the screen *L*. The limb in the U-tube *G* nearest the platinum tube contains anhydrous cupric sulphate, and the forward limb anhydrous cuprous chloride. *H* contains dried, not fused, calcium chloride, and in the bulb next to *G* is a small wad of cotton-wool. *G* and *H* are called the purifying train, and when the apparatus is not in use they should be detached from the tube and the ends closed with pieces of glass rod. The object of the cuprous chloride is to absorb any chlorine that may come over during the combustion. If any should come over, it would be mixed with hydrochloric acid and moisture, and all then would be absorbed by the salts in the tube *G*. If the carbonaceous residue is properly washed it will be found necessary to renew the salts in *G* only after it has been used for 25 or 30 combustions.

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Before making a combustion, or series of combustions, the tube should be well burned out by heating it to redness, and passing a current of oxygen through the apparatus, heating the

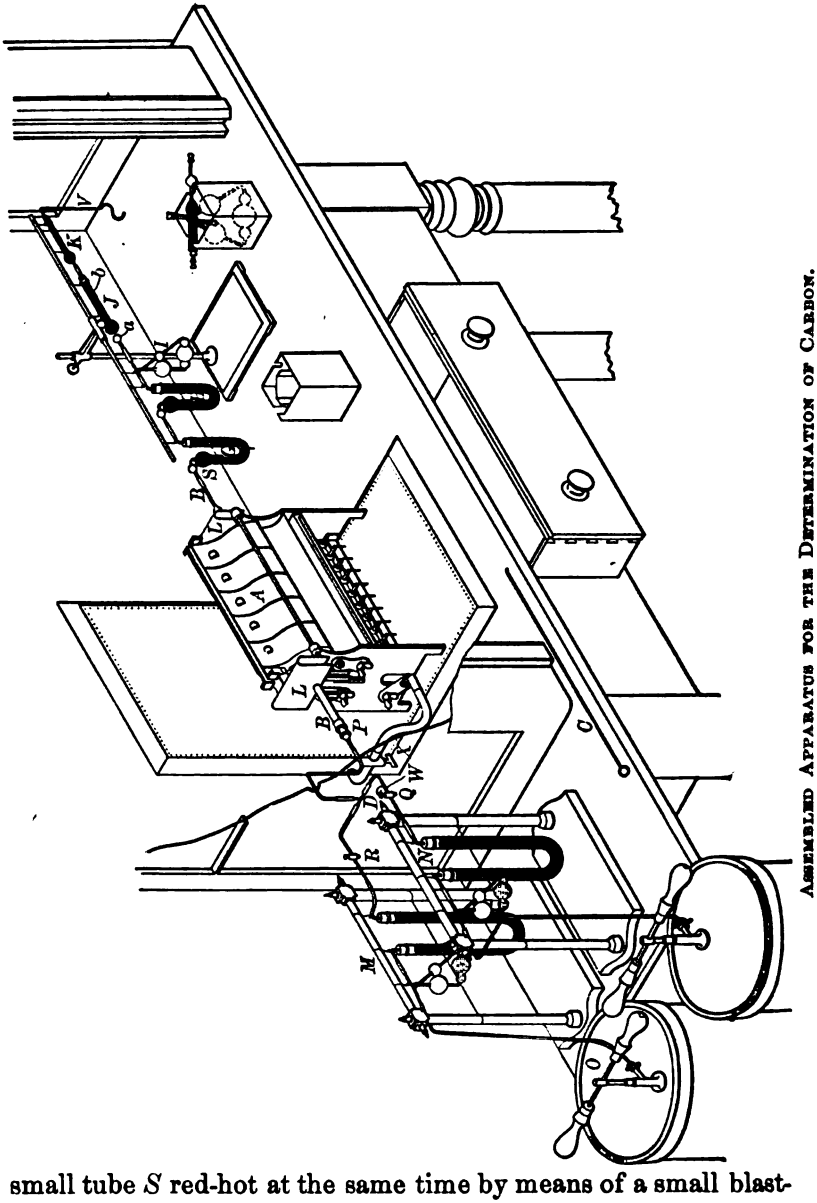

FIG. 7. ASSEMBLED APPARATUS FOR THE DETERMINATION OF CARBON.

small tube *S* red-hot at the same time by means of a small blast-lamp or Bunsen burner. During this operation fumes of sulphuric acid issue from the end of *S*, and usually, when the flame

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of the lamp is carried out to this point, it is colored green, showing that a small amount of copper salt is also volatilized. In making a combustion the platinum should not be heated above a low-red, as at a high temperature platinum becomes permeable by carbonic acid. The cylinder *O* contains oxygen under pressure, the other contains atmospheric air, which is forced through the apparatus to replace the oxygen at the end of the operation.

The purifying apparatus *M* and *N*, for oxygen and air respectively, consist of Liebig potash-bulbs filled with solution of potassium hydrate (1.27 sp. gr.), and the U-tubes, the sides next the potash-bulbs filled with dry pumice and the other sides with calcium chloride. The glass stopcocks, *Q* and *R*, shut off the purifying apparatus on their respective sides when the oxygen or air is passing through the other set. The T-tube, *D*, connects the two sets of apparatus, the third limb passing through the glass in the side of the hood, and connecting by means of the bent glass tubes with the rubber stopper *P*, which fits in the porcelain tube *B*. All the connections are made with glass tubes joined together by rubber tubing, the ends of the glass tubing being brought close together inside the rubber. This is to avoid carrying the oxygen or air through rubber tubing, which gives off volatile hydrocarbons. The absorption apparatus consists of the Liebig bulb *I*, and the drying-tube *J*. *I* contains caustic potash (1.27 sp. gr.). It is filled by attaching a short piece of rubber tubing to one end and applying suction to it, the other end being immersed in the potash solution, which has been poured into a capsule. The end must be wiped dry with a little filter-paper, and the inside of the tube dried in the same way. When filled, the bulb should contain the solution as shown in Fig. 8. When attached to the apparatus, the gas passes first into the large bulb, and, the bulbs being inclined, the gas bubbles through the solution in the three bottom bulbs. It is fitted with a loop of platinum wire, as shown in Fig. 8. The drying-tube *J* contains dried calcium chloride. The small tube *a*, Fig. 7, contains a plug of cotton-wool, and another plug of the same material is inserted after the calcium chloride at *b*. *K* is a safety-guard tube, to prevent moisture from getting into the tube *J* during the combustion. The short rubber tube *V* is used to draw a little air

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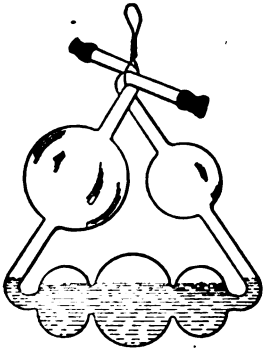
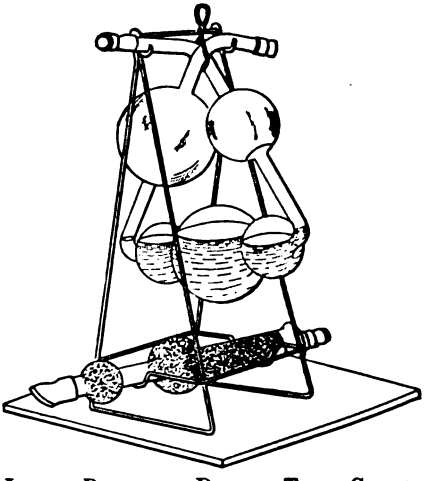
through to test the tightness of the joints. All the stoppers in the various U-tubes and drying-tubes are of rubber. The copper rod *C* is used to introduce the boats, etc., into the tube *B*, running the crooked end through the hole *W* in the glass side of the hood. When not attached to the apparatus, the ends of the potash-bulb *I* and drying-tube *J* are closed by little caps of rubber tubing (Fig. 8) made like the tips for "policemen." When on the balance, however, they should be closed with short pieces of rubber tubing containing bits of capillary glass tubing, as shown in Fig. 9. The forward end of the drying-tube is closed in the same way. These openings

FIG. 8.



LIEBIG BULB WITH SOLUTION.

FIG. 9.



LIEBIG BULB AND DRYING TUBE, CLOSED AND SUPPORTED ON WIRE STAND.

are too small to allow the condition of the atmosphere to affect the weight of the bulbs by loss or gain of moisture, but they serve to equalize the pressure and make it unnecessary to reopen the balance-case until the bulbs are weighed.

It is very necessary in filling the potash-bulb to avoid getting any of the solution on the outside of the bulb, and it is well to see that both the bulb-tube and the drying-tube are perfectly clean. Wipe the potash-bulb and drying-tube with a piece of linen, not silk (a clean linen handkerchief that does not leave lint on the glass is very good for this purpose), then place them on the balance.

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The little wire stand shown in Fig. 9 is very convenient for holding the absorption-apparatus in the balance, as it brings all the weight on the pan instead of putting the greater part on the beam alone, as is the case when the potash-bulbs are suspended from the hook on the end of the beam. Allow them to remain about 30 minutes to get the exact temperature of the balance, and weigh. Attach the absorption-apparatus as shown in Fig. 8, insert the boat in the tube by means of the rod *C*, pushing it up against the cupric oxide, insert a short roll of platinum gauze as far as the inside of the screen *L*, and close the tube with the stopper *P*. Shut the stopcocks *R* and *Q*, and, by applying suction at *V*, draw a few bubbles through the potash-bulb *I*. When the liquid recedes in the potash-bulb, it should keep its level for a few minutes; if it does not, there is a leak in some of the connections, which must be discovered and stopped before proceeding with the combustion.

When everything is tight, open *R*, and start a slow current of oxygen through the apparatus. Light the two forward burners of the furnace, turning them low to heat the copper oxide, raise the heat gradually until the tube appears red, and then light the last burner to heat the short roll of platinum gauze. As soon as this end of the tube is hot, light the third burner from the forward end, and a few minutes afterwards the fourth burner, which is directly under the forward end of the boat. Light each burner in succession from this one until all are lighted and turned high enough to heat the tube red-hot. Allow them to burn for 15 minutes, then shut off the oxygen, close *R*, open *Q*, and by means of the stopcock *T* start a current of air through the apparatus. By means of the gas-cock *X* lower all the lights of the furnace together very slowly, and finally turn them out. About 1 liter of air should run through at the rate of three bubbles a second; this will take about 25 minutes. Close *T* and *Q*, detach the absorption-apparatus, close the ends of *I* and *J* with the little rubber caps, and, after wiping the bulb and the tube gently with the linen handkerchief to remove any moisture caused by the handling, place them on the balance. Weigh with the same precautions as before; the increase in weight is carbonic acid, which contains 27.27 per cent. of carbon.

When several combustions are to be made in succession, as

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soon as the absorption-apparatus is detached as directed above, remove the boat from the tube, replace it with another containing a second sample, attach a second absorption-apparatus which has just been weighed, and proceed with the combustion. While the second combustion is in progress, the first absorption-apparatus may be weighed, and the weight then obtained can be used for the first weight of the absorption-apparatus for a third combustion. Before the absorption-apparatus shall have increased 0.5 g. in weight from the original weighing, the potash-bulb must be emptied and refilled with a fresh solution.

METHOD FOR THE DETERMINATION OF PHOSPHORUS.

BY ANDREW A. BLAIR.

This method gives an indirect determination of phosphorus by means of the estimation of the molybdic acid in the ammonium phospho-molybdate, in which form the phosphorus is precipitated. The molybdic acid is reduced to a lower state of oxidation by the reducing action of zinc and sulphuric acid, and the reduced oxide is titrated with a standardized permanganate solution, molybdic acid being again formed by the reaction.

Fig. 10 shows a form of shaking-machine for shaking four flasks at once. The construction and method of use are apparent from the sketch.

Fig. 11 shows a form of reductor which is most convenient and efficient. The tube *a* is 18 mm. in inside diameter and 300 mm. long. The small tube below the contraction with the stopcock *c* is 6 mm. in inside diameter and 300 mm. long below the stopcock. The tube is filled by placing at the point of contraction a flat piece of fine platinum gauze. On top of this is placed a plug of glass-wool, about 8 mm. thick, and then asbestos, previously treated with concentrated hydrochloric acid, thoroughly washed, ignited, and diffused in water, is poured into the reductor-tube until it forms a coating on top of the glass-wool not over 1 mm. thick. This makes a filter which prevents very small pieces of zinc or other materials from being carried through. It is necessary to clean and refill the tube from time to time, as the filter after some use becomes clogged

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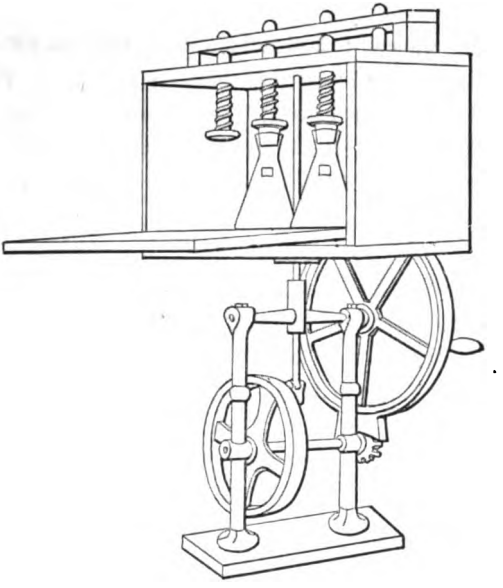
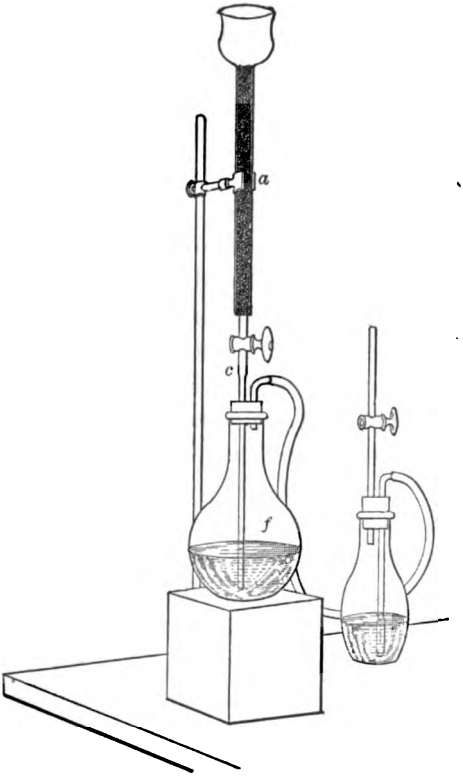
FIG. 10.



APPARATUS FOR SHAKING FLASKS.

FIG. 11.



THE REDUCTOR.

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and the liquid passes too slowly. The tube is filled, as shown in the cut, with granulated amalgamated zinc. The reductor-tube is fixed at such a height that when the block is removed from under the flask *f*, the latter may readily be detached from the tube and removed without disturbing the apparatus.

Reagents.

Nitric Acid.—Nitric acid of 1.135 sp. gr., made by mixing chemically pure nitric acid (1.42 sp. gr.) with about three parts of distilled water.

Strong Sulphuric Acid.—The chemically pure material of 1.84 sp. gr.

Dilute Sulphuric Acid.—Sulphuric acid 2.5 per cent. by volume, made by diluting 25 cc. of concentrated chemically pure sulphuric acid to 1 liter with distilled water.

Strong Ammonia.—The chemically pure material of 0.90 sp. gr.

Dilute Ammonia (0.96 sp. gr.).—Made by mixing concentrated chemically pure ammonia water of 0.90 sp. gr. with about one and one-half times its volume of distilled water.

Strong Solution of Potassium Permanganate.—For oxidizing the phosphorus and carbonaceous matter in the nitric acid solution of a steel. Made by dissolving from 12.5 to 15 g. of crystallized potassium permanganate in 1 liter of distilled water and filtering through asbestos.

Standard Solution of Potassium Permanganate.—This solution is used for titrating the reduced solutions of ammonium phospho-molybdate. Made by dissolving 2 g. of crystallized potassium permanganate in 1 liter of distilled water and filtering through asbestos. This solution is standarized as follows: Place from 0.15 to 0.25 g. of thoroughly cleaned soft steel wire, in which the iron has been carefully determined, in each of three 125-cc. Erlenmeyer flasks, and pour into each of the flasks 30 cc. of distilled water and 10 cc. of strong sulphuric acid. Cover with a small watch-glass and heat until solution is complete. Add a sufficient amount of the strong solution of potassium permanganate to oxidize the iron and destroy the carbonaceous matter, being careful to avoid an excess which would cause a precipitate of manganese dioxide. Should this occur, redissolve it by adding a very few drops of sulphurous acid and boil off every trace of the latter. Allow the solution

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in the flasks to cool and add to each 10 cc. of dilute ammonia. Pass through the reductor and titrate in the flask.

In all cases the mode of procedure in using the reductor should be as follows: Everything being clean and in good order from previous treatment with dilute sulphuric acid, and washing with distilled water, and the flask being attached to the filter-pump, pour 100 cc. of warm dilute sulphuric acid into the funnel and open the stopcock *c*. When only a little remains in the funnel forming the top of the reduction-tube, transfer the solution to be reduced to the funnel. This solution should be hot, but not boiling. Pour some of the dilute sulphuric acid into the vessel which contained the solution to be reduced to wash it, and when only a little solution is left in the funnel as before, add this to the funnel in such a way as to wash it, and follow with about 200 cc. more of warm dilute sulphuric acid, and finally with 50 cc. of hot distilled water. In no case allow the funnel to get empty; and close the stopcock *c* when there is still a little of the wash-water left in the funnel above the surface of the zinc. This precaution prevents air from passing into the reductor-tube. A blank determination is made by passing through the reductor a solution containing a mixture of 10 cc. of strong sulphuric acid, 10 cc. of dilute ammonia, and 50 cc. of water, preceded and followed by the dilute acid as described above. The amount of potassium permanganate required to give this blank a distinct color is subtracted from the amount required to give the same color to each reduced solution.

To get the value of the permanganate solution, multiply the weight of iron wire taken by the percentage of iron in the wire and divide by the number of cubic centimeters of potassium permanganate in terms of metallic iron. Multiply this result by 0.88163, the ratio of molybdic acid to iron; and the product by 0.01794, the ratio of phosphorus to molybdic acid, and the result is the value of 1 cc. of the permanganate solution in terms of phosphorus. The ratio of molybdic acid to iron given above is that found when a known amount of molybdic acid in sulphuric acid solution is passed through the reductor in the manner described above, and then titrated with potassium permanganate solution whose strength in terms of metallic iron is known. The reduction of the molybdic acid in

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the reductor in this case is to the form $Mo_{24}O_{77}$. The ratio of phosphorus to molybdic acid given above is that found by the analysis of the yellow precipitate of ammonium phospho-molybdate obtained from nitric acid solution of iron under varying conditions.

Sulphurous Acid.—A strong solution of the gas in water. Siphons of the liquefied gas may be obtained in the market.

Acid Ammonium Sulphite.—The strong chemically pure solution of the reagent diluted with 10 parts of water.

Ferrous Sulphate.—Crystals of the salt free from phosphorus.

Sodium Thiosulphate.—Crystals of the salt free from phosphorus.

These four reagents are for reducing the excess of manganese dioxide thrown down in oxidizing the carbonaceous matter in the nitric acid solutions of the steels. Any one may be used.

Molybdate Solution.—Place in a beaker 100 g. of pure molybdic anhydride, mix it thoroughly with 400 cc. of cold distilled water and add 80 cc. of strong ammonia (0.90 sp. gr.). When solution is complete, filter and pour the filtered solution slowly and with constant stirring into a mixture of 400 cc. of strong nitric acid (1.42 sp. gr.), and 600 cc. of distilled water. Add 50 mg. of microcosmic salt dissolved in a little water, agitate thoroughly, allow the precipate to settle for 24 hours, and filter before using.

Acid Ammonium Sulphate Solution.—This solution is used for washing the precipitate of ammonium phospho-molybdate. To 1 liter of water add 15 cc. of strong ammonia (0.90 sp. gr.) and 25 cc. of strong sulphuric acid (1.84 sp. gr.).

Amalgamated Zinc.—Dissolve 5 g. of mercury in 25 cc. of strong nitric acid diluted with an equal bulk of water, dilute to 250 cc. and transfer to a stout flask of about 1 liter capacity. Pour into it 500 g. of granulated zinc which will pass through a 20-mesh sieve, but not through a 30-mesh. Shake it thoroughly for a minute or two and then pour off the solution, wash the zinc thoroughly with distilled water, dry and preserve in a glass bottle for use.

Operation.

Weigh 2 g., or, in case of steels containing over 0.15 per cent. of phosphorus, 1 g. of the drillings and transfer them to

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a 250-cc. Erlenmeyer flask, pour into the flask 100 cc. of nitric acid (1.135 sp. gr.) and cover with a small watch-glass. Heat until the solution is complete and nitric oxide is boiled off. Add 10 cc. of the strong potassium permanganate solution, boil until the pink color has disappeared and manganese dioxide separates. Continue the boiling for several minutes, then remove from the source of heat and add a few drops of sulphurous acid, a small crystal of ferrous sulphate, or a solution of 0.5 g. of sodium hyposulphite in 10 cc. of water, repeating the addition at short intervals until the precipitated manganese dioxide is dissolved. Boil two minutes longer, place the flask in a vessel of cold water, or allow it to stand in the air until it feels cool to the hand, and then pour in 40 cc. of dilute ammonia (0.96 sp. gr.). The precipitated ferric hydrate will redissolve when the liquid is thoroughly mixed. When the solution is about the temperature of the hand, say 35° C., add 40 cc. of molybdate solution at the ordinary temperature, close the flask with a rubber stopper, and shake it for 5 minutes, either by the hand or in the machine, Fig. 10. Allow the precipitate to settle for a few minutes, filter on a 90-mm. filter, and wash with acid ammonium sulphate solution until 2 or 3 cc. of the wash-water give no reaction for molybdenum with a drop of ammonium sulphide. Pour 5 cc. of ammonia (0.90 sp. gr.) and 20 cc. of water into the flask to dissolve any adhering ammonium phospho-molybdate and then pour it on the precipitate in the filter, allowing the filtrate to run into a 250-cc. Griffin beaker. Wash out the flask and wash the filter with water until the solution measures about 60 cc. Add to the liquid in the beaker 10 cc. of strong sulphuric acid and pass it through the reductor exactly in the manner described for the solution of ferric sulphate in standardizing the solution of potassium permanganate, being careful to keep the end of the small tube of the reductor just below the surface of the liquid in the flask. By adding the strong sulphuric acid to the ammoniacal solution immediately before passing it through the reductor it is heated sufficiently by the chemical action to insure thorough reduction. In washing be careful that no air passes into the reductor; and when the water has been drawn through, leaving a little still remaining in the funnel, close the stopcock, detach the flask *F*, wash off the drawn-out portion of

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the reductor-tube into it, and titrate the solution with the standard permanganate. The reductor should be so arranged that the whole reduction occupies about 3 or 4 minutes. The solution that passes through should be bright-green in color. In adding the permanganate, the green color disappears first, and the solution becomes brown then pinkish-yellow, and ultimately colorless. Continue the addition of the permanganate drop by drop, shaking the flask vigorously until the solution assumes a faint pink coloration, which remains after standing one minute. Subtract from the reading of the burette the amount given by a blank determination, obtained exactly as described under the method given above for standardizing the permanganate solution, multiply the number of cubic centimeters so obtained by the value of 1 cc. in terms of phosphorus, multiply by 100 and divide by the weight taken, and the result is the percentage of phosphorus in steel.

Example.

Wire of the weight of 0.1745 g. requires 50 cc. of permanganate to give the required color. A blank determination gave 0.1 cc., so that the wire actually required 49.9 cc. of permanganate. The wire contained 99.87 per cent. of iron, then $0.1745 \times 0.9987 \div 49.9 = 0.0034923$, or 1 cc. of permanganate equals 0.0034923 g. of metallic iron. Then multiplying the value in iron by the ratio of molybdic acid to iron 0.88163 or 0.85714; and the product by the ratio of phosphorus to molybdic acid 0.01794; we have $0.0034923 \times 0.88163 \times 0.01794 = 0.000055238$, or 1 cc. of permanganate = 0.000055238 g. of phosphorus. Again, the precipitated ammonium phosphomolybdate from 2 g. of steel required 35.6 cc. of permanganate less blank 0.1 cc. = 35.5 cc.; $35.5 \times 0.000055238 \times 100 \div 2 = 0.098$ per cent. of phosphorus.

Notes and Precautions.

It will be observed that the method given above oxidizes the phosphorus in the iron by means of nitric acid, completes and perfects this oxidation and possibly neutralizes the effect of the carbon present by means of potassium permanganate, and then separates the phosphoric acid from the iron by means of molybdic acid. The molybdic acid in the yellow phospho-molyb-

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date is subsequently determined by means of potassium permanganate, the phosphorus being determined from its relation to the molybdic acid in this precipitate. The method given above applies to steel and wrought iron, but it is not yet recommended for pig-iron.

It is hardly necessary to say that all the chemicals and materials used in the analysis are assumed to be free from impurities that will injuriously affect the result.

Nitric acid of a sp. gr. of 1.135 apparently oxidizes the phosphorus just as successfully as a stronger one, while by its use solution is sufficiently rapid, and there is less trouble during the subsequent filtration due to silica.

The boiling of the solution to remove nitrous acid and assist the action of the oxidizing permanganate seems to be essential. Some steels may not require 10 cc. of the permanganate, and some, like washed metal high in carbon, may require even more. It is essential that enough should be added to cause a precipitation of manganese dioxide and give a strong pink color to the solution. This color gradually disappears on boiling. Less is required if the permanganate is added in small successive portions. Boiling two minutes after reducing the manganese dioxide removes any nitrous acid that may be formed by that operation.

In washing the yellow precipitate it shows some disposition to crawl up to the top of the filter. Care should therefore be taken to have the filter fit the funnel so closely that even if the precipitate does crawl over the top it will not be lost while washing the filter completely to the top. It is very easy to leave enough molybdic acid in the top of the filter, even though the washings are tested, to cause an error of 0.005 per cent. in the determination.

It is best to make up molybdate solution rather frequently. It is also best to keep it in the dark, at a temperature not above 28° or 30° C. Much of the so-called molybdic acid of the market is ammonium molybdate, or molybdate of some other alkali. This fact cannot be ignored in making up the molybdate solution. A series of experiments with various molybdic acids and alkaline molybdates obtained in the market indicates that if the amount of molybdic acid in the solution is that called for by the formula, irrespective of whether this amount is furnished

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by pure molybdic acid or by any of the commercial molybdates referred to, the result will be much nearer the truth than if this is not done. Good molybdic acid is the best, but the alkaline molybdates can be used. The amount of molybdic acid in these molybdates can readily be determined by dissolving 0.1 g. in 60 cc. of water to which 10 cc. of dilute ammonia have been added, filtering, adding 10 cc. of strong chemically pure sulphuric acid, and passing through the reductor as above described. The method given in the example above enables the amount of molybdic acid to be determined.

If the molybdic acid as obtained in the market, or the ammonia used in dissolving it, contains any soluble silicates, the resulting molybdate solution will be yellowish in color, and the determination made with this solution will be high, owing apparently to ammonium silico-molybdate being dragged down with the phospho-molybdate. Treatment of the molybdate solution with microcosmic salt, as described, overcomes this difficulty and gives a perfectly colorless molybdate solution. A molybdate solution tinged with yellow should never be used. It will be observed that the molybdate solution recommended above contains much less ammonium nitrate than is given by many of the formulæ now in use. Experience shows that a molybdate solution made on this formula keeps much better than those containing more ammonium nitrate. It will also be observed that the amount used for each determination is less than many methods employ. It is believed that the amount recommended is sufficient, and that the ammonium nitrate required to assist the formation of the yellow precipitate is furnished by the 40 cc. of dilute ammonia added to the nitric acid solution of the steel. Of course, the molybdate solution recommended above cannot be used for other work interchangeably with that made on the older formulæ, on account of lack of ammonium nitrate.

The directions in regard to the reductor, both as to making and use, should be strictly followed. By the use of the stopcock and the amalgamated zinc, and by keeping a little liquid in the funnel, the same blank can be obtained from a reductor almost continuously, even though two or three days of standing intervene between the blanks. It is, however, always advisable to treat with dilute sulphuric acid and wash before using, even

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though only one night has elapsed since the last previous use. If the solution to be reduced does not contain enough acid, or is not warm enough, the reduction will not be complete. Care should therefore be taken not to allow too much mixing of the dilute sulphuric acid with the solution to be reduced, either in the funnel or in the reductor-tube itself. The best asbestos to be used is the mineral known as actinolite, but any fibrous mineral which will act as a filter and not be dissolved by the acids used may be employed. Glass-wool alone will not do, as a good filter is essential in order that neither small particles of zinc nor impurities in the zinc may be drawn down into the flask with the reduced solution. The consumption of zinc is very small.

In testing the ammonium sulphide to see whether the washing of the yellow precipitate is complete, good results are obtained by putting two or three drops of yellow ammonium sulphide into a few cubic centimeters of distilled water, and allowing the washings to drop into this solution from the stem of the funnel. If iron is present in the washings it will show while the solution is still alkaline. By allowing the washings to continue running into the ammonium sulphide solution it soon becomes acid, when molybdenum, if any is present, shows by a more or less brownish color. If the acid solution is pure white from separated sulphur, the washing is complete.

Since the acidity of the solution in which the yellow precipitate is formed has an influence on its composition, it is quite desirable that the specific gravity of the nitric acid (1.135) and of the ammonia (0.96) should be taken with some care. The temperature at which the figures are correct is 15° C. It is best to use the Westphal balance in determining these gravities, but, failing this, a sufficiently delicate hydrometer may be employed.

With the amalgamated zinc in the reductor made as above described, the blank generally uses up about 0.1 cc. of the standard permanganate solution.

If the amount of sulphurous acid or other reagent added to the nitric acid solution to remove the precipitated binoxide of manganese is insufficient, a brown stain will be left on the filter-paper after the yellow precipitate is dissolved in ammonia. This may occur even though the nitric acid solution looks clear, and as no harm can arise from a slight excess of the re-

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ducing agent, it is usually more satisfactory to add an excess. It is not desirable to add the reducing agent to the solution while boiling, as under such circumstances if frequently boils over.

It is rather essential to use the dilute sulphuric acid warm, so that the general temperature of the reductor may be kept up.

A good method for cleaning the wire in standardizing the potassium permanganate is as follows: Take a round lead-pencil and make a hole in it with a pin near the end. Insert the end of the wire in this hole, and revolve the pencil until there are two turns of the wire around it. Hold the turns firmly in one hand and cut the wire so that about 0.75 mm. will remain attached to the pencil. Draw this wire through a piece of folded fine sandpaper. Seize the wire near the pencil with the filter-paper and cut off the part which was wound around the pencil and remove it. Then insert the end of the cleaned wire in the hole and revolve the pencil with one hand, holding the wire in the filter-paper in the other hand, until it is all wound loosely on the pencil. Push the coiled wire off the end of the pencil. It is now in a convenient form for weighing. (This is the method adopted by the sub-committee on standards, of the United States members of the International Steel Standards Committee. The sub-committee consisted of Messrs. Barba, Blair, Drown, Dudley (chairman), and Shimer. The sub-committee found by exhaustive experiments of specially prepared samples containing as much as 0.18 per cent. of arsenic that no arsenic was precipitated with the phosphorus when the temperature of the solution was kept below 40° C.)

TABLE II.—*Results Obtained by Andrew A. Blair.*

	H. 971.	M. 76.	H. 739.	L. 422.
Carbon Determinations.				
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Sample as received	1.414	1.040	0.434	0.098
Fine	1.421	1.039	0.451	0.105
Coarse	1.409	1.036	0.413	0.089
Average—Fine and Coarse	1.412	1.038	0.432	0.097
Phosphorus Determinations.				
Sample as received	0.019	0.018	0.022	0.026
Fine	0.020	0.016	0.022	0.025
Coarse	0.018	0.018	0.021	0.027
Average—Fine and Coarse	0.019	0.017	0.021	0.026

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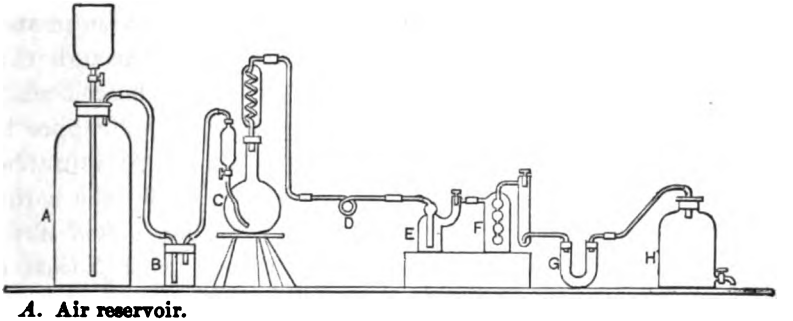
Under the heading, "Sample as received," are given the results of the analyses of the samples, the proper proportion of coarse and fine having been calculated with the eye. The samples were then sieved, the two portions "Fine" and "Coarse" weighed, and under the heading of "Average of Fine and Coarse" are given the results calculated from the proportion of "Fine" and "Coarse" found.

Method for the Determination of Carbon.

BY GUNNAR DILLNER.

Apparatus.

Fig. 12.



A. Air reservoir.
B. Bottle containing caustic potash (KOH) for absorbing CO_2.
C. Combustion vessel provided with a separate feeding orifice and a close-fitting condenser.
D. Platinum tube.
E. Tube fitted with anhydrous chloride of calcium ($CaCl_2$).
F. Absorption-tube containing caustic potash.
G. Tube filled with anhydrous chloride of calcium ($CaCl_2$).
H. Aspirator.

Apparatus for the Determination of Carbon, Used by Dillner.

Principle of Method.

The steel sample is dissolved in a solution of copper sulphate ($CuSO_4$), and the liberated carbon oxidized by means of chromic acid (CrO_3) and sulphuric acid (H_2SO_4), and the hydrocarbons formed being burnt into carbonic acid (CO_2) and water (H_2O) by making the gas, together with a current of air, pass through a platinum tube heated to redness. After passing from *D* the gas is dried by passing through a tube filled with anhydrous calcium chloride ($CaCl_2$), it then enters a tared

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absorption-tube containing caustic potash (KOH), where the carbonic acid is absorbed; an additional tube with anhydrous chloride of calcium is provided for absorbing any water which may escape from the CO_2 absorption-tubes.

On the other side of the absorption-tubes there is another tube containing anhydrous calcium chloride ($CaCl_2$) to prevent the passage of extraneous moisture into the absorption-tubes.

Manner of Procedure.

The steel sample is put into the flask *C*, 1 g. in the case of steels containing above 0.8 per cent. of carbon, 3 g. if the steel contains between 0.7 and 0.8 per cent., and 5 g. if below 0.7 per cent. of carbon. In order to expel the impure air, a current of purified air is first passed from the air cylinder through the whole apparatus. From 50 to 75 cc. of copper sulphate, according to the weight of the sample, are added through the mouth of the combustion flask *C*. This is slowly heated until the bulk of the iron is dissolved, and the sulphate of copper is discolored, an operation which will take about 20 minutes. There are then added 100 cc. of chromic acid and the same quantity of sulphuric acid. The solution is then boiled until there is no further evolution of gases. During this process a current of air is constantly passed through the apparatus in order to oxidize any hydrocarbons as they pass through the heated platinum tube. This current should give about 20 to 30 bubbles per minute when passing through the vessel *B*. The combustion will generally be accomplished within about 45 minutes. In order to make certain of all the carbon being oxidized, and also to expel from the acid solution any dissolved CO_2, there are now added 2 or 3 cc. of hydrogen peroxide (H_2O_2), which causes a rapid evolution of oxygen gas. After this addition the mixture is kept boiling for 10 minutes, when the test will be at an end and the absorption-tube with contents ready for weighing. From the increase in weight due to CO_2 the percentage of carbon is calculated.

Reagents Required.

Concentrated solution of sulphate of copper.
Concentrated solution of chromic acid.
Sulphuric acid (1 concentrated + 1 water).

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Method For the Determination of Phosphorus.

BY GUNNAR DILLNER.

Steel to the weight of 1.64 g. is dissolved in nitric acid (sp. gr. 1.22). The solution is boiled until the sample is dissolved, when it is evaporated to dryness, and then strongly heated on an iron slab for one hour, after which it is again dissolved, in hydrochloric acid, and once more evaporated to dryness, this being done in order to render the silica insoluble. The residue is once more dissolved in 20 cc. of nitric acid, and to accelerate the process, if need be, a few drops of hydrochloric acid added. The silica is now filtered off, and there is added to the filtrate an equal quantity of a solution of molybdate of ammonia. The beaker containing this solution is then placed into a water-bath at a temperature of 40° C., where it is left to stand for four hours in order to make the phospho-molybdate settle out. The precipitate thus obtained is then collected on a tared filter and weighed, every milligramme of the weight of the precipitate representing 0.001 per cent. of phosphorus in the sample.

In order to obtain the tare and weight of the precipitate, the filter is weighed at first after having been dried from one to one and a half hours in a drying-oven, and then left to cool for 10 minutes in a desiccator, and the same proceeding is then repeated again after filtration, the filter with the precipitate being also submitted to similar operations of drying and desiccating.

Report By Gunnar Dillner on the Various Analytical Methods.

Carbon Determinations.

The methods of estimating carbon used in this investigation can be referred to as the "direct" or "indirect." When the solution of the sample and the oxidation of the carbon takes place within the same vessel, the method is a direct one; when, on the contrary, the steel solution is filtered, and the carbon residue is burnt off in a special furnace, the process can be named an indirect one. The "Stead" direct process, as well as the "Saernstroem" method, belong to the former, whilst

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the "Stead" and "Blair" potash-copper process and the "Jüptner" mode of procedure are of the indirect class.

The methods employed in the following determinations were the Saernstroem for the carbon, modified in certain points by myself, and for phosphorus the Eggertz method, the details of which were rigidly adhered to.

TABLE III.—*Results Obtained by Gunnar Dillner.*

Method Employed.	H. 971.	M. 76.	H. 739.	L. 422.
	Carbon Determinations.			
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Stead's direct process	1.49	1.01	0.45	0.10
	1.49	1.04	0.43	0.09
	1.41	1.01	0.46	0.10
	1.45	1.02	0.46	0.10
Average	1.46	1.02	0.45	0.10
Stead's potash-copper process	1.41	1.02	0.36	0.12
	1.27[a]	0.98	0.45	0.08
	1.41	.....	.....	.....
	1.44	.....	.....	.....
Average	1.42	1.00	0.41	0.10
Jüptner's method	1.30	0.96	0.40	0.09
	1.42	0.94	0.49	0.10
	1.41	.....	0.42	.....
	1.38	.....	0.37	.....
Average	1.38	0.95	0.42	0.10
Saernstroem's method	1.43	1.00	0.43	0.09
	1.45	1.00	0.45	0.10
	1.46	1.02	0.44	0.10
Average	1.45	1.01	0.44	0.10
	Phosphorus Determinations.			
Stead's method	0.021	0.026	0.026	0.031
	0.022	0.026	0.025	0.031
Eggertz's method	0.023	0.022	0.022	0.030
	0.022	0.023	0.024	0.030

[a] This was excluded in averaging the results, as combustion was probably not complete. If it is included the average is 1.42 per cent.

On examining, then, the two indirect methods, we find that they are similar, in so far as the steel is dissolved in both cases by means of a copper solution, the only difference being in detail of working. According to the "Jüptner method," solu-

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tion of the steel is effected without heating, but is facilitated by stirring, the time required being from 2 to 3 hours, whereas, according to Stead's indirect process, the solution is boiled and the sample more rapidly dissolved (in about 20 minutes). Now, in my opinion, the main weak point common to the indirect methods is in the way the dissolving is effected. As is well known, and has been pointed out on several occasions by various authorities,[1] there is always an evolution of hydrocarbon gases, when steel is dissolved in copper chloride, which escape. In that way I account for the lower results obtained by the indirect processes. It also seems that this evotion of hydrocarbon gases will be more intense the higher the carbon is in the steel, and what is more important, according to my experience in the present research, the same steel has been proved to yield a larger amount of hydrocarbon gases after hardening than was evolved when in an unhardened state. As a proof of this statement I may mention that by testing according to Stead's indirect method a type of Fagersta open-hearth steel in the hardened and unhardened state, the following results were obtained: Unhardened steel, 0.92, and steel hardened at 850° C., 0.78 per cent. of carbon.

It thus appears that the application of either of the indirect methods will necessitate hardened steels being annealed previous to being tested for carbon.

Again, the filtering-operation characteristic of the indirect methods not only requires the utmost care and attention so as to avoid a more or less considerable loss of carbon, but it also involves a great waste of time. The conclusion to be drawn seems to be that neither of these methods ought to be considered as equal to the direct methods from the point of view of practical convenience, accuracy, or reliability of the results obtained, owing mainly to the constant source of errors pointed out above, namely, the unavoidable loss of carbon caused by the evolution of hydrocarbon gases taking place during the dissolving process.

As to the direct methods, one will find from the comparative

[1] A. Lebebur, *Prüfung der Zuverlassigheit der gebrauchlichsten Verfahrungsweisen zur Bestimmung des in Eisen enthaltenen Kohlenstoffs, Verhandlungen des Vereins zu Beförderung des Gewerbfleisses*, 1893, vols. vi.–vii. ; also C. G. Saernstroem, *Jernkontorets Annaler*, 1884.

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results obtained in Table III. that the most satisfactory results have been obtained by means of the "Stead direct process" and the "Saernstroem method," yielding not only the highest, but also the most concordant results.

A rather important point in favor of the "Stead direct process," as compared with the Stockholm method, is the unquestionable saving of time offered by the former, which only requires half an hour, while the corresponding estimation by means of the other could hardly be achieved in less than from 1.75 to 2 hours. It seems to me that these two methods give equally accurate results, and that with the exception mentioned above it would hardly be justifiable to proclaim the precedence of either.

When extending the application of these methods so as to include estimation of carbon in pig-iron, the aspect of the matter becomes somewhat different. While being aware that the object of this committee is to test the various methods with regard to their application to steel, it has occurred to me in the course of the present researches, that the value of a method as such also depends upon its having a more or less general application, and that it might thus be of a certain interest to take this opportunity of undertaking some further comparative tests with materials other than steel.

On analyzing pig-iron according to the Saernstroem method, I have repeatedly stated that the carbon contained therein, the graphite as well as the combined carbon, will be completely oxidized in about 2 hours. In order to ascertain this, I have used the method of intercepting the gas-current every now and then by means of a pinch-cock, and then weighing the absorption-tube until there was no further increase in the weight. During the last few years about 30 analyses, in duplicate, of pig-iron have been made at the testing laboratory of the Royal Technical University in Stockholm by this method, and satisfactory results have always been obtained.

The material selected on this occasion, to serve the purpose of testing the value of Mr. Stead's direct process for estimating carbon in pig-iron, consisted of three samples of common pig-iron, A, B and C, together with one sample of ferrochrome (containing 52.4 per cent. of chromium).

The percentage of carbon according to the Saernstroem method gave the following results:

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	Per Cent.	Per Cent.
Pig-iron, A, . . .	Carbon = 2.72	(Graphite = 2.51)
Pig-iron, B, . . .	Carbon = 2.93	(Graphite = 2.76)
Pig-iron, C, . . .	Carbon = 2.86	(Graphite = 2.68)
Ferrochrome, . . .	Carbon = 3.52	.....

Before attempting the application of Mr. Stead's direct process to such material, some preliminary experiments were undertaken, by means of which it was ascertained that the sample, when being too strongly heated, caked on the surface and thus prevented the complete combustion of the carbon, and that consequently such excessive heating ought to be avoided; in fact, the carbon obtained by means of this process in the case of sample "A," after four hours of strong heating, was only 1.51 per cent., whereas a subsequent analysis of the same pig-iron "A," at a lower temperature, gave the following results,—

	Per Cent. of Carbon.
After 10 minutes, ...	1.34
After ½ hour, ...	2.17
After 1½ hours, ...	2.20
After 2½ hours, ...	2.45

By using a pulverized sample, there was at last obtained, in the case of this pig-iron "A," 2.62 per cent. of carbon.

The following data will afford a more convenient survey of the comparative results obtained while applying the two direct methods to this material,—

Description of Material.	Carbon by Saernstroem's Method.	Carbon by Stead's Direct Process.	
		Combustion, 2½ Hours.	Combustion, 8 Hours.
	Per Cent.	Per Cent.	Per Cent.
Pig-iron, A.....	2.72	2.62	.....
Pig-iron, B.....	2.93	2.01	.....
Pig-iron, C.....	2.86	2.57	2.57
Ferrochrome.....	3.52	0.67	0.70

On examining the comparative results, it will be seen that Mr. Stead's direct process, however excellent and superior it has proved to be as a method of estimating carbon in steel, could hardly, in its present form, be acknowledged as a suitable method for estimating the carbon in pig-iron, while, on the

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other hand, the Saernstroem method has, in fact, proved to be able to give equally satisfactory results in the case of quite different materials, such as pig-iron and even certain special alloys of steel which are very difficult to dissolve.

Phosphorus Determinations.

Regarding the phosphorus tests there are only a few remarks to make.

The results of these tests are given in Table III., from which it will be found that the methods used by Baron Jüptner and Mr. Blair have here been omitted. Only the other two—that described by Mr. Stead, and the Eggertz method adopted at the testing laboratory of the Royal Technical University in Stockholm—were considered sufficient to serve the purpose of the present researches. The reason for this will be easily understood, for on comparing the detailed descriptions of the Jüptner, Blair and Stead methods, it will be seen that they are practically the same excepting as regards the mode and manner of ascertaining the weight of the precipitate, a difference so slight as to have no influence on the results obtained.

From the results obtained by the two methods, it is evident that one gives as good results as the other, but it ought to be acknowledged that the comparative rapidity of the process used by Mr. Stead when arsenic is absent constitutes an essential advantage.

In order to ascertain whether arsenic, when present, interferes with the determination of phosphorus in steel, another series of comparative tests were made, in which arsenic acid, equivalent to 0.05 per cent. of the weight, was added to the solution of the steel. The results of these tests were as follows,—

	H. 971.	M. 76.	H. 737.	L. 422.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Stead's method.....	0.020	0.024	0.022	0.030
Eggertz's method.....	0.020	0.022	0.024	0.029

On applying the Stead method, every particular and precaution, as prescribed in the author's memorandum on estimating phosphorus in steel when arsenic is present, were care-

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fully followed, whereas in the Eggertz method the only special precaution strictly observed was that the precipitation of the phospho-molybdate was effected at exactly 40° C.

By means of separate tests it was also found that at higher temperatures than 50° C. there were quite appreciable amounts of arsenic precipitated together with the phosphorus.

Methods for the Determination of Carbon in Steel.

BY J. E. STEAD.

The system adopted in the laboratory of Messrs. Pattinson and Stead, Middlesbrough, for the determination of carbon in metals and in steel, has been that of direct combustion in oxygen on the one hand, and combustion of the residue after dissolving in potassium cupric chloride on the other. The method adopted for the last 25 years has been the latter process, and is in principle identical with that described by Mr. Blair.[2] There are, however, differences in detail which it is important should be noted.

Instead of the ordinary combustion-furnace we have adopted the very excellent gas muffle-furnace supplied by Messrs. Griffin & Co. The body has openings at each side of it so that several tubes can be placed one over the other through the furnace. The muffle itself is removed and the opening is closed by the fire-brick doors. Two furnace bodies are placed one above the other. By working in this way with one gas-burner it is possible to heat a many as eight tubes and to conduct eight combustions at one time.

The oxygen is distributed to the several tubes, but is first purified by passing through potash and then through strong sulphuric acid.

The tubes themselves are of porcelain, and are 20 in. long and 0.75 in. in diameter. They are fitted with India-rubber corks through which pass glass tubes. Plugs of copper oxide are placed in the combustion-tubes so as to ensure the oxidation of any CO gas which might be formed during the combustion. After the gases leave the tubes they are passed

[2] *The Chemical Analysis of Iron.* Philadelphia, 5th ed., 1902.

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through U-tubes containing respectively anhydrous copper sulphate, cuprous chloride, chromic acid, and calcium chloride, and are then passed through tubes, previously weighed, containing soda-lime, or through the ordinary potash-bulbs. These absorb the CO_2 produced.

Indirect Combustion.

In analyzing steel or iron, 2.727 g. of the material are placed in a No. 4 beaker, and upon this is poured 200 cc. of a solution of potassium cupric chloride (90 g. potassium chloride, 40 cc. of strong hydrochloric acid, per liter). The liquid is gently warmed so as to produce circulation, and is finally boiled until the steel is dissolved and no trace of free copper remains. The carbon residue is then filtered through asbestos, and is thoroughly washed with water containing a little hydrochloric acid, and finally with pure distilled water. A little asbestos is pressed upon the surface of the carbon residue, so as to make a sandwich with asbestos on each side and the residue in the center. The asbestos with residue is removed from the funnel and is pressed into a platinum boat 3 in. long, $\frac{3}{8}$ in. wide and $\frac{3}{8}$ in. deep. The boat with contents is placed in a water-bath to dry, and when completely free from water is ready to place in the combustion-tube.

The furnace having been previously lit and the tubes raised to a temperature of about 900° C., the potash absorbing-tubes are carefully weighed and a current of oxygen passed through the whole range of the apparatus during a period of 10 minutes. Without removing the oxygen in the potash apparatus, they are weighed and again connected up and oxygen passed through for a further 10 minutes and are again weighed. The weights should remain constant and show neither any increase nor decrease. Should there be an increase in weight there should be an investigation made to discover the cause. Should there be a decrease, it is an indication that the calcium chloride drying-tubes at the rear of the potash-bulbs require renewing. If there is no increase or decrease in weight the potash-bulbs are again connected. The India-rubber cork and oxygen connection are removed from the combustion-tube and the boat containing the residue is placed therein and is rapidly, with one stroke of a piece of glass rod, pushed into the heated area. The glass rod is withdrawn and

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the India-rubber cork inserted, an operation which does not occupy more than two seconds. The oxygen gas is turned on and kept flowing until after repeated weighings of the potash absorption-bulbs there is no increase in weight. The CO_2 thus obtained is calculated into carbon. The increase in weight of the bulbs in milligrammes indicates the number of hundredths of carbon present in the sample of steel.

At the end of the combustion, by means of a hooked wire, the boat with the residual asbestos is withdrawn from the tube and a second boat containing carbon residue inserted.

The furnace is not allowed to cool between the combustions.

In a single tube it is possible to make at least 10 combustions in 6 hours, and if there are eight tubes in a furnace it is possible to make about 80 analyses per day.

As a matter of fact, in the furnace employed at Middlesbrough only four tubes are placed in the furnace.

Direct Combustion Method.

In making a determination of carbon by this process it is absolutely necessary that the turnings should not exceed 0.5 mm. thick. When the drillings have a greater thickness than this it is advisable to use the cupric-potassium chloride process already described. The drillings may be either in the form of shavings or in fine powder; preferably the former. When being burnt, theoretically every particle should be isolated from its fellow so that the oxygen in passing over the material may act on every side of them. In practice this is impossible, but a near approach to it can be effected by mixing the drillings with granulated particles of crushed magnesite brick or with granulated calcined manganese-ore. When the drillings consist entirely of spiral pieces there is no need for any admixture whatever, as the oxygen can then act upon all parts without the intervention of any mechanical support. If, however, the particles of steel or metal are in fine powder, it is imperative that this mechanical suspension should be effected.

The boats are made by cutting asbestos sheets into strips about 4 in. long and 0.75 in. wide, and covering them with china-clay on one side, with the aid of a spatula. The strips thus prepared are molded into boat form by means of the fingers. They are then dried and finally calcined in a muffle-

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furnace, after which treatment they are ready for use. One boat is required for each combustion.

A mixture of 2.727 g. of steel with the necessary proportion of granulated magnesite brick or manganese oxide is placed in a boat and the latter is at once put into the heated combustion-tube and the metal burned in oxygen. If the furnace is sufficiently hot, complete combustion should not take more than half an hour.

The results obtained, by working the two methods above referred to, agree most closely, and it has been proved, after three years' constant use of both processes worked side by side, that if the proper precautions are taken the direct combustion-process gives most accurate results.

It is necessary at this point to refer to an article written by Mr. Lawrence Duffy.[3] In this paper evidence is given as to the value of the direct combustion-process, and the method he describes is almost identical with that I have given.

Although Mr. Duffy's proofs of the value of the method are possibly the first to be published, it is nevertheless a fact that for three years previous to the publication of the paper, it was in constant use in our own laboratory, and also in one or two of the largest metallurgical steel-works in this country. They are, however, notwithstanding what I have stated, of the very greatest value, as they show that in Sheffield, as in Middlesbrough and in other prominent steel-works, the direct combustion-process, if properly carried out in the manner described, is a simple and accurate one of determining the carbon in steel, and one that takes so little time that it should, in all our steel-works in the near future, entirely take the place of the color method, which is well known is liable to give results more than 30 per cent. out of the truth.

Methods for the Determination of Phosphorus.

BY J. E. STEAD.

The method employed for the determination of phosphorus is that known by the name of "The Molybdic Acid Process," in which the phospho-molybdate of ammonium after precipita-

[3] *Chemical News*, June 19, 1903.

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tion, is weighed, and the amount of phosphorus calculated from the weight obtained.

In practice we use two modifications of this method, one in which the steel is dissolved in nitric acid solution (1.2 sp. gr.) decolorized by potassium permanganate, and, after the addition of a sufficient quantity of ammonium nitrate, the phosphorus is precipitated with ammonium molybdate and the precipitate weighed on a tared filter-paper. This system is very useful for steels containing practically no arsenic and silicon.

In the other method precautions are taken to separate any arsenic which might be present: 4.89 g. of the steel are dissolved in 35 cc. of nitric acid of 1.42 sp. gr. and 25 cc. of hydrochloric acid. The solution is evaporated to dryness, taken up with hydrochloric acid, a little water is added, and afterwards pure granulated zinc in quantities sufficient to reduce completely the ferric chloride to ferrous chloride. When the excess of zinc is dissolved, a few drops of ammonium sulphide are added and the solution is violently agitated. If the black iron sulphide is not completely dissolved, a little more hydrochloric acid must be added until solution is effected. Arsenic sulphide is precipitated at once and is coagulated on shaking, and may be filtered off after vigorous agitation or after standing over-night. After filtering, the residue contains the silica and arsenic sulphide, the filtrate the phosphorus. The filtrate is oxidized with nitric acid after boiling off the free hydrogen sulphide, and the bulk of liquid is reduced to 70 cc. by evaporation. Nitric acid is then added to oxidize the iron, strong ammonia until the solution is just neutral, then 8 cc. of the same ammonia in excess. Nitric acid is now added until the hydrated iron oxide has just passed into solution, and a final addition of 5 cc. of nitric acid is added. If the volume of liquid is greater than 100 cc., it may be evaporated until reduced to that bulk. When boiling, 20 cc. of a 10 per cent. solution of ammonium molybdate in water are added, and the beaker with contents well shaken and allowed to stand on the table to allow the ammonium phospho-molybdate to completely separate. When the supernatant liquid is perfectly bright and the precipitate has settled to the bottom, the solution is filtered through tared filter-papers. These latter are made in the following manner, viz.: Two folded papers after thoroughly drying in a

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water-bath are placed on opposite pans of an accurately adjusted balance. From the heavier paper portions are clipped off the apex until one filter counterpoises the other. They are then returned to the water-bath for from 15 to 20 minutes. They are then taken out and final and accurate adjustment made. The whole filter is placed inside the other and in this the precipitate is collected.

After washing the precipitate with water containing 1 per cent. of nitric acid it is given three washings with distilled water. The filter with contents is then dried at about 110° C.

After drying thoroughly the filters are separated, the counterpoise paper being placed on one pan and the paper with precipitate on the other. Weights are added to the pan with the counterpoise paper until equilibrium is established. The weight thus obtained, divided by three, gives the exact percentage of phosphorus present in the original metal.

After considerable number of experments it has been found in our laboratory that it not infrequently happens that all the phosphorus is not precipitated when the liquid is at a temperature of 40° C. and under.

TABLE IV.—*Results Obtained by J. E. Stead.*

Carbon Determinations.				
Method Employed.	H. 971.	M. 76.	H. 739.	L. 422.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Direct combustion	1.44	1.035	0.42	0.10
Variations	0.03	0.02	0.02	0.01
Averages	1.44	1.035	0.42	0.10
Solution in cupric-potassium chloride, and combustion of residue—				
Variations	0.03	0.04	0.02	0.01
Averages	1.40	1.02	0.42	0.096
Baron H. von Jüptner's method—				
Averages	1.375	1.015	0.436	0.088
Saernstroem's method—				
Variations				
Averages	1.404	1.033	0.425	0.093
Phosphorus Determinations.				
Method used in Mr. Stead's laboratory—				
Variations	0.002	0.002	0.002	0.002
Averages	0.0230	0.0245	0.0245	0.0295

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We find, furthermore, that on precipitating at a higher temperature than 40° C., arsenic, if present, is liable to come down with the phosphorus precipitate. The amount varies considerably, and the exact law which determines the quantity so precipitated has not been fully ascertained. For these reasons we consider it advisable invariably to separate the arsenic in the first instance, and then to precipitate at a sufficiently high temperature to ensure complete precipitation of the phosphorus.

As there was not sufficient of the standard drillings to make further investigations, trials were made with Eggertz's method on drillings from other samples of steel similar to those experimented upon by the Committee, the results obtained being as follows:

	Percentage of Phosphorus.			
	No. 1.	No. 2.	No. 3.	No. 4.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Eggertz's method.....	0.021	0.022	0.024	0.022
Stead's method.....	0.022	0.022	0.025	0.023

Report by J. E. Stead.

The results obtained in my laboratory lead to the following conclusions on the relative value of the several methods of analysis:

Carbon Determinations.

Method of Baron H. von Jüptner.—This indirect wet combustion-process yielded lower carbon results than any of the other methods.

Method of Saernstroem.—This is a direct wet combustion-process and yielded results closely concordant with the percentages of carbon obtained by the dry combustion method. Theoretically there is no objection to this method. The precaution taken to pass the gases evolved mixed with air through a red-hot tube effectually prevents the escape of any hydrocarbons.

The Direct Combustion Method.—The direct combustion of steel in a current of oxygen gas has long been in use, but until the last few years it was considered necessary to mix the steel drillings with copper oxide, lead chromate, lead oxide, or other

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oxidizing substances, previous to burning it in the combustion-tube. Since oxygen, perfectly free from impurities, has been commercially obtainable, experiment has shown that it is not usually necessary to make any such admixture, and that if the temperature of the furnace is sufficiently high, and the drillings of steel not too fine or too coarse, combustion may be effected rapidly and completely, the whole of the carbon being obtained as carbonic anhydride.

My attention was drawn to the value of this method by one of my old pupils who had used it with success in a large steel-works in England in 1890. In my opinion this method holds the premier position.

The Indirect Combustion Method.—This method has long been in use, and is recognized as the standard method in England and America.

The results obtained by its use in this investigation were rather lower than those obtained by the direct methods, but in general practice such variations are unusual.

For more than three years it has been the practice in our laboratory, whenever possible, to test samples of steel received for analysis by both the indirect and direct dry combustion-processes. The results have, almost without exception, closely agreed.

The copper chloride method described so fully by Mr. Blair is practically the same as that used in Middlesbrough, but the system for mechanically agitating the cold cupric liquor adopted by Mr. Blair is an improvement on the method I have used, and one which should be generally adopted.

It will be noted that the results obtained by Mr. Blair closely agree with those obtained in Middlesbrough.

Grey pig metals in the form of drillings are invariably tested by the cupric-potassium chloride method. The direct process is unsuitable for the determination of carbon in such metals.

Determination of Phosphorus.

All the methods employed in this research are modified varieties of the molybdic acid method.

Baron H. von Jüptner's Method.—This method is practically identical with the Eggertz method, the only difference being in the mode of weighing the precipitate.

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No trials were made by the special method of weighing adopted by Baron H. von Jüptner.

Eggertz's Method.—The Eggertz method of determining phosphorus appears to give good results, but in past experience it has been found, more particularly in the case of pig-irons, that occasionally the phosphorus is not completely precipitated by heating to 40° C., and that therefore it is liable to give lower results than the truth. There is also the objection that the solution containing the phosphorus has to stand for many hours at a temperature of 40° C. to allow the precipitate to separate.

The Pre-Separation of Arsenic Method.—When arsenic is absent the phosphorus may be precipitated at between 80° and 100° C., and it all falls out of solution in less than one hour.

Although more laborious, it has been found that it takes less time to obtain a result if arsenic is first separated than when following the method of Eggertz.

Many years ago it used to be a common practice to filter the phospho-molybdate upon smooth English filter-papers, to brush the precipitate off when quite dry, and weigh it separately; but such a procedure invariably gave low results, because it was impossible to remove the whole of the precipitate from the pores of the paper.

The method adopted by Mr. Blair and the American chemists of dissolving the precipitate from the paper in ammonia and reducing the solution afterwards, and determining the molybdenum volumetrically, and the method by Messrs. Brearley and Ibbotson of determining the molybdenum in the precipitate by precipitation with a salt of lead, are very excellent and free from objection; but when one has 20 or 30 phosphorus determinations to make per day, the extra manipulation required is very considerable, as compared with weighing the precipitate directly upon properly tared filters.

It seems to be incontrovertible that if the original precipitate is capable of being accurately weighed there is no need whatever to make the process more complicated.

The method of weighing on counterpoised filters has been found to be accurate. If there is any objection to weighing the filter-papers the precipitate can be weighed in Gooch crucibles, or in glass filter-tubes suitably arranged with asbestos plugs at their bases.

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Review of the Work Done.

The following table gives the averages of the results obtained by working with the methods approved by each analyst, viz.:—

Table V.—*Averages by Methods Approved by Each Analyst.*

	H. 971.	M. 76.	H. 789.	L. 422.
Carbon Determinations.				
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Dillner	1.46	1.02	0.45	0.10
	1.45	1.01	0.44	0.10
Jüptner	*a* 1.57	1.105	0.459	0.141
	b 1.52	1.071	0.439	0.0936
	c 1.439	1.046	0.424	0.140
	d 1.450	1.068	0.459	0.142
Blair	1.415	1.040	0.434	0.098
	1.412	1.038	0.432	0.097
Stead	1.44	1.035	0.420	0.10
	1.40	1.020	0.420	0.096
Phosphorus Determinations.				
Dillner	0.0225	0.0225	0.0230	0.030
Jüptner	0.0305	0.0320	0.0275	0.027
Blair	0.0190	0.0180	0.0220	0.026
Stead	0.0230	0.0245	0.0245	0.0295

Baron H. von Jüptner has not tested the samples by the method he handed to the committee, and does not give any opinion as to what method he considers best.

Neither Baron H. von Jüptner nor Mr. Blair have employed the direct dry combustion-process.

On comparing the above results it will be seen that the greater part of the determinations agree closely, but there are exceptions to this rule.

With regard to the methods for the determination of carbon, Mr. Dillner and Mr. Stead both agree in strongly recommending the two direct combustion-processes. It will, however, be noticed that excepting in high-carbon steel the results obtained by the cupric chloride indirect method agree very closely with those obtained by the direct processes.

Theoretically, the wet and the dry combustion-processes should give the most trustworthy results.

With regard to the phosphorus determinations, the results are again somewhat variable.

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Mr. Blair's results are generally lower than those obtained by the other analysts. The results of Messrs. Dillner and Stead agree throughout very closely, whereas those obtained by Baron H. von Jüptner are, with one exception, higher than the others. Theoretically, there is no reason why they should not agree. They were all obtained by the molybdate process, Mr. Blair having determined the molybdenum precipitated volumetrically, whilst Baron H. von Jüptner adopted a method which slightly differs from that of Messrs. Dillner and Stead in the system of weighing.

The committee are continuing their investigations, and are endeavoring to ascertain whether, or not, hydrocarbon gases do escape when steel is dissolved in cupric-potassium chloride, and the reasons for the discrepancies shown above, the results of which will be presented on some future occasion.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Development and Use of High-Speed Tool-Steel.*

BY J. M. GLEDHILL.

INTRODUCTORY.

IT would doubtless have been felt by many but a few years back that there was little left to be said on the subject of crucible tool-steel, and that something akin to finality had been arrived at in its manufacture and general treatment. Probably such feeling was justifiable when it is remembered that the making of steel in crucibles is by far the oldest method known, dating back from time immemorial, it being indeed impossible to accurately trace its origin and earliest development; but it seems certain that carbon-steel was made and used thousands of years ago for cutting-tools. Proof of this may be seen by the the marvelous carvings and workings on the intensely hard stone-work of the ancients, for it would be difficult to conceive by what means, other than with steel tools, such work could have been executed; and it is wonderful to contemplate that steel cutting-tools should have been used so long ago, whilst the principle of manufacturing them—that is, by fusion of iron and charcoal in crucibles—was then in a measure on the same lines as we work on at the present day. Archæologists have discovered that the Chinese made steel in crucibles long before the Christian era.

"Wootz" steel fabricated in India centuries ago was crucible steel, as was also the celebrated Damascus steel, produced at the forges of Toledo, and, curiously, this latter steel furnishes yet another proof that "There is nothing new under the Sun," for it is recorded that Damascus steel contained certain percentages of tungsten, nickel, manganese, etc., some of the very elements, in fact, contained in the present modern high-speed steel, so that a latent high-speed steel may be said to have existed

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centuries ago, and all that was necessary to bring out its inherent powers would have been the heating of it in a "paradoxical" manner, so to speak; that is, to such a high degree of temperature as was long thought would impair or destroy the nature of such steel. When, therefore, we look back on the period for which crucible steel has been known to the world's history, some may not unnaturally think that there has been time enough to have fully fathomed its mysteries, leaving little more to be said on the subject. It is then all the more remarkable that a discovery, made but a few years back, and which has since revolutionized the treatment of crucible tool-steel, should have remained so long a hidden secret.

A very important advance was made 30 or 40 years ago, when "Mushet," or self-hardening steel, was introduced. This was the valuable invention of Robert Mushet, who, after a long series of experiments, made whilst he was manager of the Titanic Steel Co., succeeded in producing a tungsten steel, and its introduction was a great advancement on the cutting powers of ordinary crucible steel, and for many years "Mushet" steel held a foremost place amongst tool-steels.

It is now to America, however, that all honor must be given for the next great step in having "set the pace" and led the way in the present remarkable advancement in tool-steel, and the author would here like to record that the greatest credit is due to Messrs. Taylor & White, who, at the Bethlehem Steel-works of America, initiated high-speed cutting, and at the exhibit of their firm in Paris some years back, what were then considered to be astonishing results in speeds of cutting-steel were publicly demonstrated. Since then, still greater developments have been made by the author's firm in high-speed steels, and with increased experience in its manufacture, treatment and application in our workshops, results in cutting-powers far beyond expectation have been attained.

In those branches of the engineering world necessitating the use of tool-steels in course of manufacture of their products, there is probably no section of greater importance thereto than that of the application and practical use of what is now known as "high-speed steel," and it may be said without doubt that no development in the annals of metallurgy has been more striking than the production of such steel; whilst the alacrity

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with which users have up to now appreciated and adopted high-speed cutting steel may be said to be almost comparable with the rapid powers of the steel itself! Perhaps this is not surprising when we look back and reflect that for many years past, previous to the advent of high-speed steel, practically but little advance had been made in the cutting-powers of tool-steels, feeds and speeds remaining more or less in a normal condition, and it must undoubtedly have often appeared to users generally that the cutting-powers of the ordinary tool-steels were very slow, and that the turn-out of work could be greatly enhanced and economized if the cutting-powers of tool-steels could be substantially improved by some research of the metallurgist. Time has eventually realized those hopes to a large extent, and hence the desire now on all sides to use high-speed steel wherever possible. When it is seen that, with high-speed tools, steel can now be turned and machined at a rate up to 500 ft. per min., and cast-iron drilled at 25 in. per min., it must be admitted that this is an astonishing advancement on the cutting-speeds of from 30 to 50 ft. per min. with ordinary crucible steel.

The production of steel capable of such cutting-powers has not been obtained without exhaustive scientific research and trial, and the author may state that experiments have been made by him extending over a period of four years, during which time some 80 different compositions of high-speed steels were produced, and hundreds of trials made with them by actual cutting, and whilst some excellent results were obtained with many of them, others gave but indifferent ones.

In the manufacture and production of high-speed steels (the best being made by the crucible process) the author has proved conclusively that the most satisfactory results are obtained only by using the purest qualities of Swedish or Dannemora irons, which, on account of their freedom from impurities, render them most suitable for producing tool-steels that will best retain their cutting-edges, and also the use of the highest qualities of the various alloys and other ingredients employed in the composition of the steel.

Special care is required in the melting and subsequent treatment of ingots alloyed with high percentages of other metals so as to ensure homogeneity and regularity of quality, which

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is one of the most important points to be considered, for no permanent advantage or economy can be relied on if good results are obtained from one bar of steel and inferior results from the next, as will be the case where attention and experience are not exercised.

Summary of Results of Experiments.

The high-speed steels of the present day are combinations of iron and carbon with—

(1) Tungsten and chromium.
(2) Molybdenum and chromium.
(3) Tungsten, molybdenum and chromium.

These present many interesting varieties and offer a wide field for research. The author has made a large number of experiments to ascertain comparative cutting-powers of steels produced by varying proportions of these elements, and it may be interesting to briefly state the results of some of those investigations.

Influence of Carbon.—A number of tool-steels were made with the carbon percentage varying from 0.4 to 2.2 per cent., and the method of hardening was to heat the steel to the highest possible temperature without destroying the cutting-edge, and then rapidly cooling in a strong air-blast. By this simple method of hardening it was found that the greatest cutting-efficiency is obtained where the carbon ranges from 0.4 to 0.9 per cent., and such steels are comparatively tough. Higher percentages are not desirable, because great difficulty is experienced in forging the steels, and the tools are inferior. With increasing carbon-content the steel is also very brittle, and has a tendency to break with unequal and intermittent cutting.

Influence of Chromium.—Having thus found the best carbon-content to range from 0.4 to 0.9 per cent., the next experiments were made to ascertain the influence of chromium varying from 1 to 6 per cent. Steels containing a low percentage are very tough, and perform excellent work on the softer varieties of steel and cast-iron, but when tried on harder materials the results obtained were not so efficient. With an increased content of chromium the nature of the steel becomes much harder, and greater cutting-efficiency is obtained on hard materials. It was

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observed with an increase of chromium there must be a decrease in carbon to obtain the best results for such percentage of chromium.

Mention may here be made of an interesting experiment to astertain what effect would be produced in a rapid steel by substituting vanadium for chromium. The amount of vanadium present was 2 per cent. The steel readily forged, worked very tough, and was hardened by heating to a white heat and cooling in an air-blast. This tool when tried on medium steel stood well, but not better than the steel, with the much cheaper element of chromium in it.

Influence of Tungsten.—This important element is contained in by far the greater number of the present high-speed steels in use. A number of experiments were made with the tungsten-content ranging from 9 to 27 per cent. From 9 to 16 per cent., the nature of the steel becomes very brittle, but at the same time the cutting-efficiency is greatly increased, and about 16 per cent. appeared to be the limit, as no better results were obtained by increasing the tungsten beyond this figure. Between 18 and 27 per cent. it was found that the nature of the steel altered somewhat, and, instead of being brittle, it became softer and tougher, and whilst such tools have the property of cutting very cleanly they do not stand up so well.

Influence of Molybdenum.—The influence of this element at the present time is under investigation, and our experiments with it have so far produced excellent results, and it is found that where a large percentage of tungsten is necessary to make a good rapid steel, a considerably less percentage of molybdenum will suffice. A peculiarity of these molybdenum steels is that in order to obtain the greatest efficiency they do not require such a high temperature in hardening as do the tungsten steels, and if the temperature is increased above 1,000° C. the tools are inferior and the life shortened.

Influence of Tungsten with Molybdenum.—It was found that the presence of from 0.5 to 3 per cent. of molybdenum in a high tungsten steel slightly increased the cutting-efficiency, but the advantage gained is altogether out of proportion to the cost of the added molybdenum.

Influencc of Silicon.—A number of "rapid" steels were made with silicon-content varying from a trace up to 4 per cent. Sil-

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icon sensibly hardens such steels, and the cutting-efficiency on hard materials is increased by additions up to 3 per cent. By increasing the silicon above 3 per cent., however, the cutting-efficiency begins to decline. Various experiments were made with other metals as alloys, but the results obtained were not sufficiently good by comparison with the above to call for comment.

An analysis of one of the best qualities of rapid steels produced by the author's firm is as follows:—("*A. W.*" *Steel*) carbon, 0.55, chromium, 3.5, and tungsten, 13.5 per cent.

What may be said to determine a high-speed steel, as compared to an ordinary tool-steel, is its capability of withstanding the higher temperatures produced by the greatly increased friction between the tool and the work due to the rapid cutting. An ordinary carbon-steel containing, say, 1.2 per cent. of carbon when heated slightly above the critical point and rapidly cooled by quenching in water becomes intensely hard. Such a steel gradually loses this intense hardness as the temperature of friction reaches, say, 500° F. The lower the temperature is maintained the longer will be the life of the tool, so that the cutting-speed is very limited. With rapid-cutting steels the temperature of friction may be greatly extended, even up to from 1,100° to 1,200° F., and it has been proved by experience that the higher the temperature for hardening is raised above the critical point and then rapidly cooled, the higher will be the temperature of friction that the tool can withstand before sensibly losing its hardness. The high degree of heating (almost to melting-point, in fact) which is necessary for hardening high-speed steel forms an interesting study in thermal treatment and is indeed a curious paradox, quite inverting all theory and practice previously existing. In the case of hardening ordinary carbon-steels very rapid cooling is absolutely necessary, but with high-speed steels the rate of cooling may take a considerably longer period, the intensity of hardness being increased with the quicker rate of cooling.

In his admirable paper on rapid steel for tools,[1] M. Le Chatelier says "that steel undergoes at 700° C. a change of nature which has been studied in all its details by M. Osmond. This

[1] *Bulletin de la Société D' Encouragement pour L' Industrie Nationale.*

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transformation, like a great number of chemical transformations, takes place with more or less considerable delay according to certain other circumstances. When heating, the transformation will take place above 700° C.—for example, from 750° to 800° C., according to the rapidity of heating. When cooling, below 700° C., the quickness with which this transformation takes place at a given temperature is governed by a general law of chemical phenomena, and this rapidity is so much the greater (1) As the absolute temperature in question is highest; (2) As it is at its greatest distance from the point of transformation."

It may be added that the transformation at the critical point takes only a very short time in the case of simple carbon-steels and the influence of such elements as chromium, tungsten, molybdenum, vanadium, and manganese is to considerably retard this change.

Turning now to some points in the heat-treatment of high-speed steel, one of the most important is the process of thoroughly annealing it after working into bars. Accurate annealing is of much value in bringing the steel into a state of molecular uniformity, thereby removing internal strains that may have arisen, due to casting and tilting, and, at the same time, annealing renders the steel sufficiently soft to enable it to be machined into any desired form for turning-tools, milling-cutters, drills, taps, screwing-dies, etc.

The annealing of high-speed steel is best carried out in muffle-furnaces designed for heating by radiation only, a temperature of 1,400° F. being maintained from 12 to 18 hours according to the section of the bars of steel dealt with.

Further advantage also results from careful annealing by minimizing risks of cracking when the steel has to be reheated for hardening. In cases of intricately-shaped milling-tools having sharp square bottom-recesses, fine edges, or delicate projections, and on which unequal expansion and contraction are liable to operate suddenly, annealing has a very beneficial effect towards reducing the cracking to a minimum.

Increased ductility is also imparted by annealing, and this is especially requisite in tools that have to encounter sudden shocks due to intermittent cutting, such as planing- and slotting-tools, or others suddenly meeting projections or irregularities on the work operated on.

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The following tensile tests of "A. W." high-speed steel in the normal, annealed, hardened, and hardened and then tempered states, also the microscopic views given in Figs. 1, 2, 3, 4 and 5 of the steels in the states described, will be instructive as showing the effects of the heat-treatment.

No. of Specimen.	Condition.	Elastic Limit.	Breaking Strain.	Elongation.	Contraction in Area.	Remarks.
		Tons per Sq. In.	Tons per Sq. In.	Per Cent.	Per Cent.	
1	Normal.	.....	112.0	.....	.....	.....
2	Annealed.	40.0	58.0	18.0	35.0	Fibrous.
3	Hardened.	.....	62.0	.....	.....	.....
4	Hardened and Tempered.	.....	89.0	.....	.....	.....

The test-pieces of the above are shown on the table.

It will be observed that the ductility of the annealed specimen is very good, rendering the steel in a condition to withstand the great pressures due to the forces thrown upon it when cutting.

In preparing high-speed steel ready for use, the process may be divided principally into three stages: Forging, hardening, and grinding. It is, of course, very desirable that high-speed steel should be capable of attaining its maximum efficiency and yet only require treatment of the simplest kind, so that an ordinarily skilled workman may easily deal with it, otherwise the preparation of tools becomes an expensive and costly matter, and materially reduces the advantages resulting from its use. Fortunately, the treatment of the rapid steel produced by the author's firm is of the simplest; simpler, in fact, than ordinary carbon-steels or the old self-hardening steels, as great care had to be exercised in the heating of the latter steels, for if either were heated above a blood-red heat, say 1,600° F., the danger of impairing their efficency by burning was considerable; whereas, with the high-speed steel, heating may be carried to a much higher temperature, even up to melting-point, it being practically impossible to injure it by burning. The steel may be raised to a yellow heat for forging, say 1,850° F., at which temperature it is soft and easily worked into any desired form, the forging proceeding until the temperature lowers to a good red heat, say 1,500° F., when work on it should cease and the steel be reheated.

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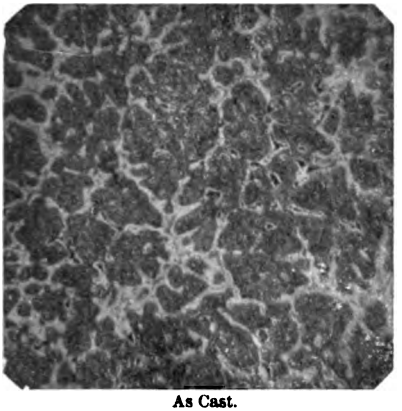
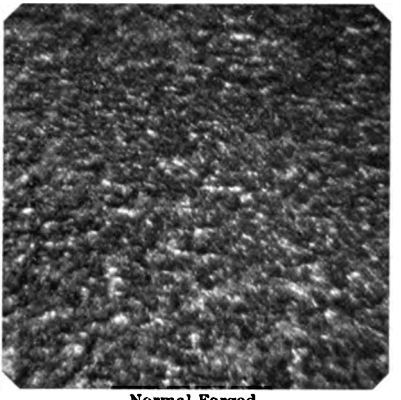
FIG. 1.



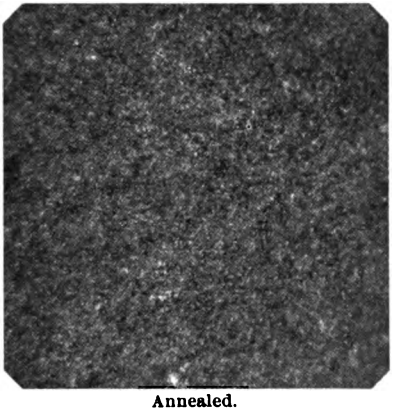
As Cast.

FIG. 2.



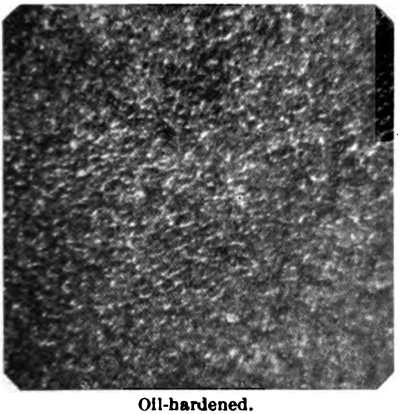
Normal Forged.

FIG. 3.



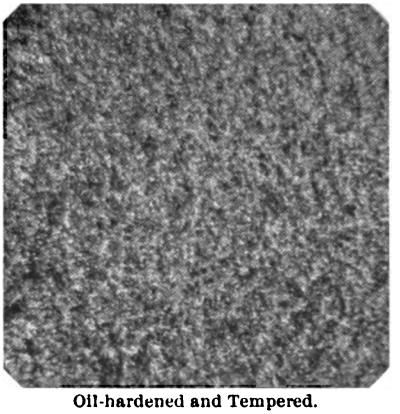
Annealed.

FIG. 4.



Oil-hardened.

FIG. 5.



Oil-hardened and Tempered.

MICRO-PHOTOGRAPHS OF SPECIMENS OF "A. W." HIGH-SPEED STEEL.

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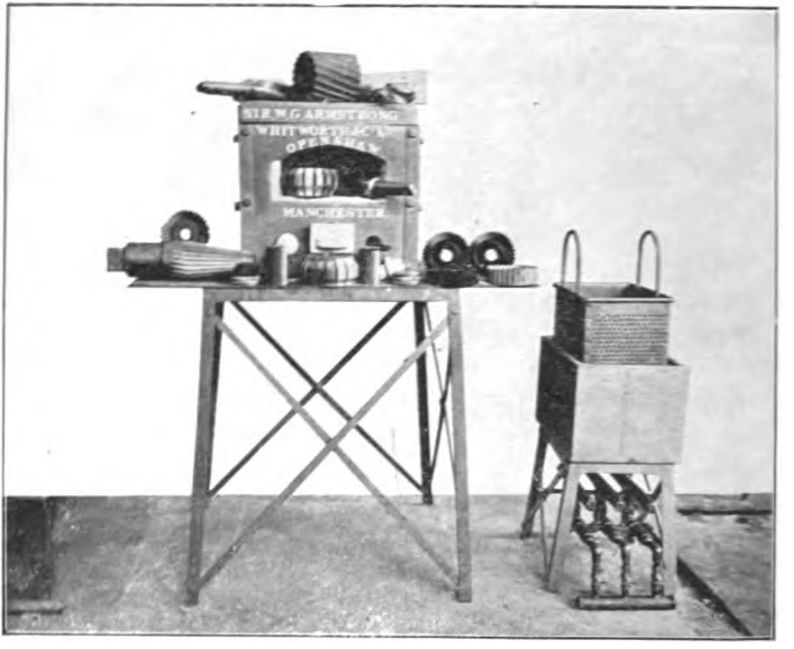
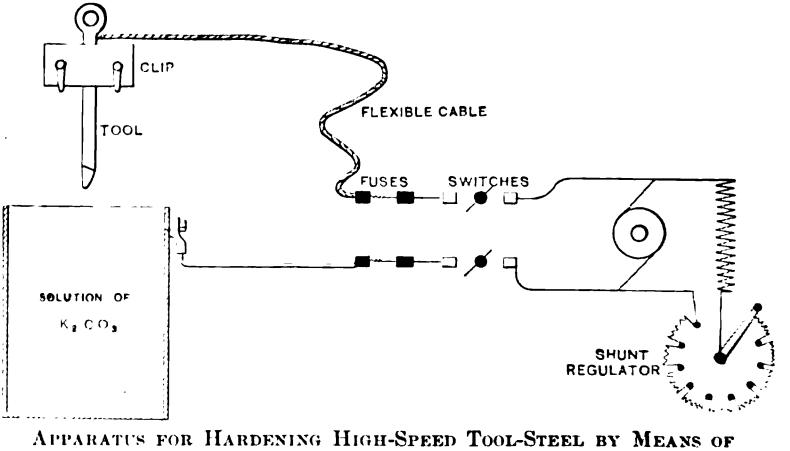
FIG. 6.



MUFFLE FURNACE FOR HARDENING MILLING-CUTTERS, ETC., MADE OF HIGH-SPEED STEEL; ALSO TANK AND CAGE FOR AFTERWARDS TEMPERING THEM IN OIL. (SIR W. G. ARMSTRONG, WHITWORTH & CO., LTD., OPENSHAW, MANCHESTER.)

FIG. 7.



APPARATUS FOR HARDENING HIGH-SPEED TOOL-STEEL BY MEANS OF AN ELECTRIC ARC SHOWING COOLING-TANK.

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In heating a bar of high-speed steel preparatory to forging (which heating is best done in a clear coke-fire) it is essential that the bar be heated thoroughly and uniformly, so as to ensure that the heat has penetrated to the center of the bar, for if the bar be not uniformly heated, leaving the center comparatively cold and stiff, whilst the outside is hot, the steel will not draw or spread out equally, and cracking will probably result. A wise rule in heating is to "hasten slowly."

It is not advisable to break pieces from the bar whilst cold, the effect of so doing tending to induce fine end-cracks to develop which ultimately may extend and give trouble, but the pieces should be cut off whilst the bar is hot, then be reheated as before and forged to the shape required, after which the tool should be laid in a dry place until cold.

The temperature for hardening high-speed steel varies somewhat according to the class of tool being dealt with.

When hardening turning-, planing-, or slotting-tools, and others of similar class, the point or "nose" of tool only should be gradually raised to a white melting-heat, though not necessarily melted, but even should the point or the tool become to a more or less extent fused or melted no harm is done. The tool should then be immediately placed in an air-blast and cooled down, after which it only requires grinding and is then ready for use.

Another method which may be described of preparing the tools is as follows:—

Forge the tools as before, and when quite cold grind to shape on a dry stone or dry emery-wheel, an operation which may be done with the tool fixed in a rest and fed against the stone or emery-wheel by a screw, no harm resulting from any heat developed at this stage. The tool then requires heating to a white heat, but just short of melting, and afterwards completely cooling in the air-blast. This method of first roughly grinding to shape also lends itself to cooling the tools in oil, which is specially efficient where the retention of a sharp edge is a desideratum,—as in finishing-tools, capstan and automatic lathe-tools, brass-workers' tools, etc.

In hardening where oil-cooling is used, the tools should be first raised to a white heat, but without melting, and then cooled down either by air-blast or in the open to a bright-red heat,

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say 1,700° F., when they should be instantly plunged into a bath of rape- or whale-oil, or a mixture of both.

Referring to the question of grinding tools, nothing has yet been found so good for high-speed steels as the wet sandstone, and the tools ground thereon by hand-pressure, but where it is desired to use emery-wheels it is better to roughly grind the tools to shape on a dry emery-wheel or dry stone before hardening. By so doing, the tools require but little grinding after hardening, and only slight frictional heating occurs, but not sufficient to draw the temper in any way, and thus their cutting-efficiency is not impaired. When the tools are ground on a wet emery-wheel and undue pressure is applied, the heat generated by the great friction between the tool and the emery-wheel causes the steel to become hot, and water playing on the steel whilst in this heated condition tends to produce cracking.

With regard to the hardening and tempering of specially formed tools of high-speed steel, such as milling- and gear-cutters, twist-drills, taps, screwing-dies, reamers, and other tools that do not permit of being ground to shape after hardening, and where any melting or fusing of the cutting-edges must be prevented, the method of hardening is as follows:—

A specially arranged muffle-furnace heated either by gas or oil is employed, and consists of two chambers lined with fire-clay, the gas and air entering through a series of burners at the back of the furnace, and so under control that a temperature up to 2,200° F. may be steadily maintained in the lower chamber, whilst the upper chamber is kept at a much lower temperature.

Before placing the cutters in the furnace it is advisable to fill up the hole and keyways with common fire-clay to protect them.

The mode of procedure is now as follows:—

The cutters are first placed upon the top of the furnace until they are warmed through, after which they are placed in the upper chamber (see Fig. 6) and thoroughly and uniformly heated to a temperature of about 1,500° F., or, say, a medium red heat, when they are transferred into the lower chamber and allowed to remain therein until the cutter attains the same heat as the furnace itself, viz., about 2,200° F., and the cutting-edges become a bright-yellow heat, having an appearance of a glazed or greasy surface. The cutter should then be withdrawn whilst

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the edges are sharp and uninjured, and revolved before an air-blast until the red heat has passed away, and then whilst the cutter is still warm—that is, just permitting of its being handled—it should be plunged into a bath of tallow at about 200° F. and the temperature of the tallow-bath then raised to about 520° F., on the attainment of which the cutter should be immediately withdrawn and plunged in cold oil.

Of course there are various other ways of tempering, a good method being by means of a specially arranged gas-and-air stove into which the articles to be tempered are placed, and the stove then heated up to a temperature of from 500° to 600° F., when the gas is shut off and the furnace with its contents allowed to slowly cool down.

Another method of heating tools is by electrical means, and by which very regular and rapid heating is obtained, and where electric current is available, the system of electric heating is quick, reliable and economical, and a brief description of this kind of heating may be of interest.

One method adopted of electrically heating the points of tools and the arrangement of apparatus is show in Fig. 7. It consists of a cast-iron tank, of suitable dimensions, containing a strong solution of potassium carbonate together with a dynamo, the positive cable from which is connected to the metal clip holding the tool to be heated, whilst the negative cable is connected direct on the tank. The tool to be hardened is held in a suitable clip to ensure good contact. Proceeding to harden the tool the action is as follows:—

The current is first switched on, and then the tool is gently lowered into the solution to such a depth as is required to harden it. The act of dipping the tool into the alkaline solution completes the electric circuit and at once sets up intense heat on the immersed part. When it is seen that the tool is sufficiently heated, the current is instantly switched off, and the solution then serves to rapidly chill and harden the point of the tool, so that no air-blast is necessary.

Another method of heating the point of tools is by means of the electric arc, the heating-effect of which is also very rapid in its action. The general arrangement and form of the apparatus here employed being as illustrated in Fig. 8; and the area of tool subjected to electric contact, in Fig. 9.

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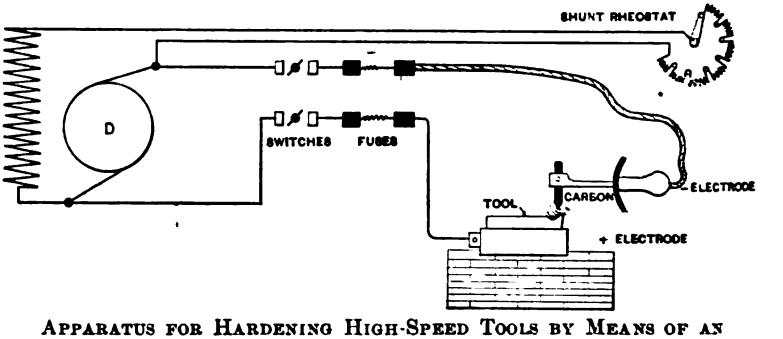
The tool under treatment and the positive electrode are placed on a bed of non-conducting and non-combustible material and the arc started gradually at a low voltage and steadily increased as required, by controlling the shunt-rheostat, care being taken not to obtain too great a heat and so fuse the end of the tool. The source of power in this case is a motor-generator consisting

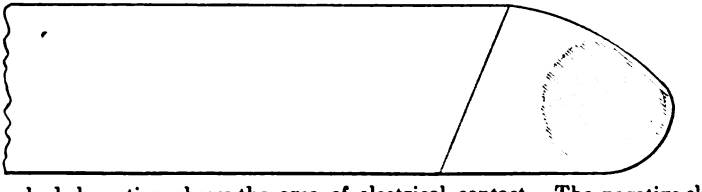
FIG. 8.



APPARATUS FOR HARDENING HIGH-SPEED TOOLS BY MEANS OF AN ELECTRIC ARC.

of a continuous-current shunt-wound motor at 220 volts, coupled to a continuous-current shunt-wound dynamo at from 50 to 150 volts. Arcs from 10 to 1,000 amperes are then easily produced and simply and safely controlled by means of the shunt-rheostat.

FIG. 9.



The shaded portion shows the area of electrical contact. The negative electrode should be kept moving over this surface without approaching too near the cutting-edge of the tool.

AREA OF TOOL SUBJECTED TO ELECTRICAL CONTACT.

Tempering.—Electricity is also a very efficient and accurate means of tempering such forms of tools as milling-, gear-, hobbing- and other similar cutters; also large hollow taps, hollow reamers, and all other hollow tools made of high-speed steel, where it is required to have the outside or cutting-portion hard, and the interior soft and tenacious, so as to be in the best con-

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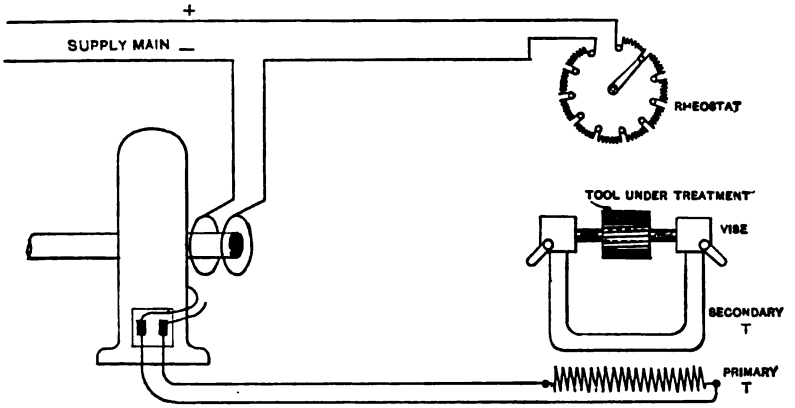
dition to resist the great stresses put upon the tool by the resistance of the metal being cut, and which stresses tend to cause disruption of the cutter if the hardening extends too deep.

By means of the apparatus illustrated in Fig. 10 this tempering or softening of the interior can be perfectly and quickly effected, thus bringing the cutter into the best possible condition to perform rapid and heavy work.

Tempering of hollow cutters, etc., is sometimes carried out by the insertion of a heated rod within the cutter, and so drawing the temper, but this is not entirely satisfactory or scientific, and is liable to induce cracking by too sudden heat-application, and further because of the difficulty of maintaining the neces-

FIG. 10.



APPARATUS FOR TEMPERING MILLING-CUTTERS, ETC., ELECTRICALLY.

sary heat and temperature required, and afterwards gradually lowering the heat until the proper degree of temper has been obtained. In electrical tempering these difficulties are overcome, as the rod is placed inside the cutter quite cold, and the electric current gradually and steadily heats up the rod until the correct temperature is reached, when it can be held at such temperature as long as is necessary, and the current can be gradually reduced until the articles operated on are cold again, and consequently the risk of cracking by too sudden expansion and contraction is reduced very greatly. The apparatus used is very simple, as will be seen by reference to the sketch. It consists of a continuous-current shunt-wound motor directly coupled to a single-phase alternating-current dynamo of the re-

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volving-field type, giving 100 amperes at 350 volts, 50 cycles per second, the exciting current being taken from the works supply-main.

The power from the alternator is by means of a stepdown-transformer, reduced to current at a pressure of 2 volts, the secondary coil of the transformer consisting of a single turn of copper of heavy cross-section, the extremities of which are attached to heavy copper bars carrying the connecting-vices holding the mandril upon which the cutter to be tempered is placed. The secondary induced current, therefore, passes through a single-turn coil, through the copper bars and vices and mandril.

Although the resistance of the complete circuit is very low, still, owing to the comparatively high specific resistance of the iron mandril, the thermal effect of the current is used up in heating the mandril which gradually attains the required temperature, slowly imparting its heat to the tool under treatment until the shade of the oxide on the tool satisfies the operator.

The method adopted to regulate the heat of the mandril is by varying the excitation-current of the alternator by means of the rheostat. An extremely fine variation and perfect heat control are easily possible by this arrangement.

Uses.—Having touched upon the development and thermal treatment of high-speed steel, it will now be opportune to refer to its practical use and to some of the most recent work done with it. It is sometimes contended that, on the whole, not much advantage or economy results from using high-speed steel, but it is easy to prove very greatly to the contrary, and the author proposes to give some figures and facts as to its use and advantage, not only by knowledge gained from results of his own firm, but also from information supplied by many important engineering-establishments as to their present workshop practice and for which he is indebted.

That great economy is effected is beyond all doubt, from whichever point of view the question is looked at; for it is not only rapidity of cutting that counts, but the output of machines is correspondingly increased, so that a greater production is obtained from a given installation than was possible when cutting at low speeds with the old tool-steel, and the work is naturally produced at a correspondingly lower cost, and of course it follows from this that in laying down new plant and

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machines, the introduction and use of high-speed steel would have considerable influence in reducing expenditure on capital account.

It has also been proved that high-speed cutting is economical from a mechanical standpoint and that a given horse power will remove a greater quantity of metal at a high speed than at a low speed, for although more power is naturally required to take off metal at a high than at a low speed (by reason of the increased work done) the increase of that power is by no means in proportion to the large extra amount of work done by the high-speed cutting, for the frictional and other losses do not increase in anything like the same ratio as a high-cutting speed is to a low-cutting speed. A brief example of this may be given in which the power absorbed in the lathe was accurately measured electrically.

Cutting on hard steel, with $\frac{3}{16}$-in. depth of cut, $\frac{1}{16}$-in. feed and speed of cutting 17 ft. per min., a power of 5.16 h.p. was absorbed, and increasing the cutting-speed to 42 ft. per min., the depth of cut and feed being the same; there was a saving in power of 19 per cent. for the work being done.

Another experiment with depth of cut $\frac{8}{8}$ in. and traverse $\frac{1}{16}$-in., compared with $\frac{1}{16}$-in. traverse and $\frac{3}{16}$-in. depth of cut, showed a saving in power of as much as 28 per cent. and still proceeding with a view of increasing the weight of metal removed in a given time the feed was doubled (other conditions being the same) and a still further saving of power resulted. In a word, as in the majority of things, so it is with rapid cutting, the more quickly work can be produced the cheaper the cost of production will be.

Again, as regards economy there is not only a saving effected on the actual machine work, but since the advent of high-speed cutting it is now possible, in many instances, to produce finished articles from plain rolled bars, instead of following the old practice of first making expensive forgings and afterwards finishing them in the machine. By this practice, not only is the entire cost of forging abolished, but the machining on the rolled bar can be carried out much quicker and cheaper in suitably arranged machines, quicker even than the machining of a forging can be done.

Many wonderful examples in proof of this are laid on the

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table, with labeled particulars of speeds and times of production from rough-rolled bars, and the saving of time is indeed remarkable. Taking the two articles illustrated below, these were machined from plain rolled bars with high-speed steel in 45 and 13 min. respectively, as against $3\frac{3}{4}$ and $1\frac{1}{4}$ hr. when made from forgings and using ordinary tool-steel.

Another remarkable sample of the gain resulting from the use of high-speed cutting from rolled bars is illustrated on Fig. 11, the articles in this case being securing-bolts, made by the author's firm, for armor-plates. Formerly, where forgings were first made and then machined with ordinary self-hardening steel, a production of 8 bolts per day of ten hours was usual. With the introduction of rapid-cutting steel, 40 similar bolts from the rolled bar are now produced in the same time, thus giving an advantage of five to one in favor of quick cutting, and also in addition abolishing the cost of first rough-forging the bolt to form; in fact, the cost of forging 1 bolt alone amounted to more than the present cost of producing to required form 12 such bolts by high-speed machining. The cutting-speed at which these bolts are turned is 160 ft. per min., the depth of cut and feed being respectively $\frac{3}{4}$ in. and $\frac{1}{32}$ in., the weight of metal removed from each bolt being 62 lb., or 2,480 lb. in a day of ten hours, the tool being only ground once during such period of work, and from such an example as this it will be at once apparent what an enormous saving in plant and costs results. On the same principle the sleeves (see appendix) for these bolts are produced from bars, 60 being made in one day of ten hours, this being even a greater saving on the old system than the bolt example shows.

An illustration of the lathe on which this work is done is also shown on Fig. 12. It is a 12-in. lathe of special design and strength for rapid and heavy cutting, and has a link driving-belt of 7.5 in. wide, running at a very high velocity and driven by its own motor, so that the power absorbed can always be observed whether the lathe is running idle or c

Equally remarkable results are obtained by operat stock bars with high-speed milling cutters, and one ex amongst many, may be cited, which is shown on Fig. 13. hexagon nuts for bolts $3\frac{3}{8}$ in. in diameter are made from bars, the cutting-speed of milling being 150 ft. per min., a

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...bolts produced per day of 10 hours. Cutting-speed, 160 ft. per min.; ...of cut, 1¼ in.; feed, ⅛ in.; weight removed from each blank, 20 lb.; total ...removed, 2,160 pounds. (Sir W. G. Armstrong, Whitworth & Co., Ltd., ...ester.)

...ING SLEEVES FOR ARMOR-PLATE BOLTS (9 TO 12 INCH HIGH-SPEED LATHE).

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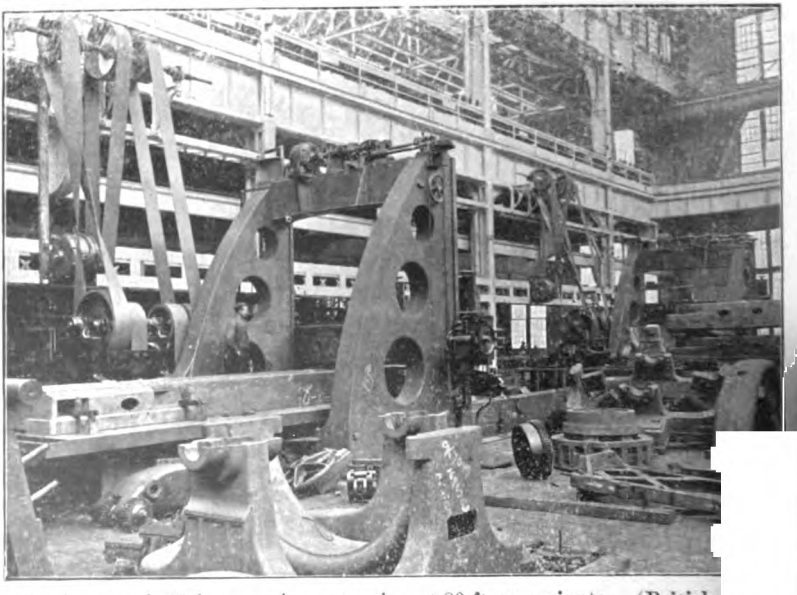
FIG. 13.

Ninety nuts are produced in a day of 10 hours. Total weight of cuttings removed per day, 675 pounds. (Sir W. G. Armstrong, Whitworth & Co., Ltd., Manchester.)

MAKING HEXAGON NUTS FROM ROLLED BARS WITH CUTTERS OF "A. W." HIGH-SPEED STEEL.

FIG. 14.



Cutting-speed, 36 ft. per min., returning at 80 ft. per minute. (British inghouse Electric and Manufacturing Co., Ltd., Manchester.)

PLANING-MACHINES FITTED WITH MITCHELL'S PATENT DRIVE, U "A. W." HIGH-SPEED STEEL.

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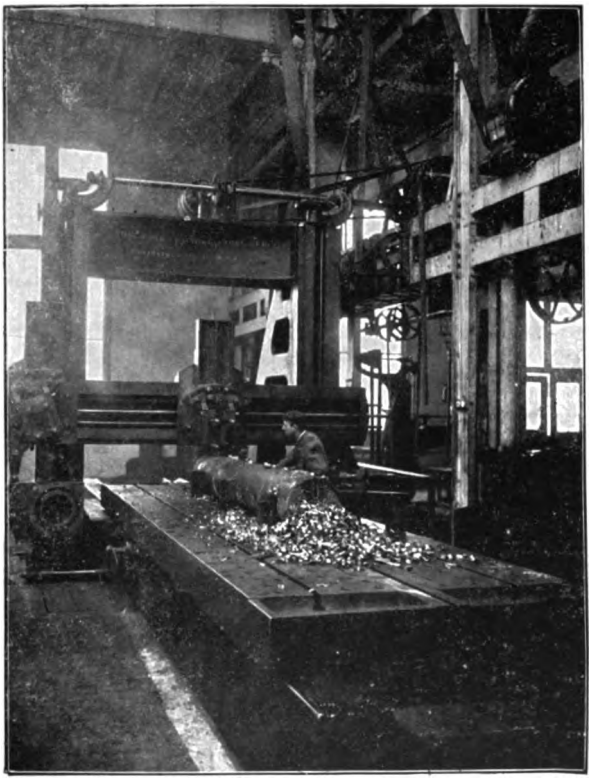
FIG. 15.



Cutting-speed, 65 ft. per min., returning 160 ft. per min.; planing 145 ft. of carbon-steel. (British Westinghouse Electric and Manufacturing Co., Ltd., Manchester.)

PLANING-MACHINE FITTED WITH MITCHELL'S DRIVE, USING "A. W." HIGH-SPEED STEEL.

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FIG. 16.

Cutting-speed, 150 ft. per min.; depth of cut, $\frac{3}{16}$ in.; feed, 16 cuts per inch of traverse. (Sir W. G. Armstrong, Whitworth & Co., Ltd., Manchester.)

FORTY-FIVE MINUTES' WORK WITH "A. W." HIGH-SPEED STEEL.

FIG. 17.

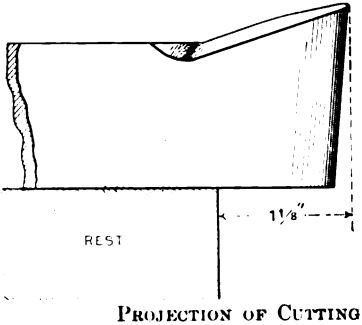
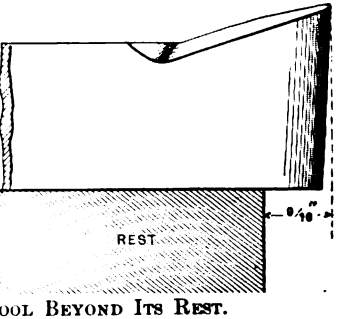


FIG. 18.



PROJECTION OF CUTTING-TOOL BEYOND ITS REST.

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a production of 90 nuts per day, against 30 formerly. More than 90 nuts could have been produced had the machine been more powerful.

A further economy of the great saving resulting from the use of high-speed steel was recently supplied to the author by one of the largest manufacturers of textile machinery in Lancashire, and they wrote as follows:—

"Our results up to the present time, without being phenomenal, have demonstrated the fact that there are great possibilities for effecting considerable saving by its use, and, as examples, would mention the following:—

"In the drilling of cast-iron rails, we have been able to increase the speed from four to eight times, and in one case of milling cast-iron brackets we have, by building a special milling-machine, and the use of the new material, increased the output six times as compared with that obtained under the old conditions.

"In another case we have been able to obtain 20 per cent. reduction in the piece-work price of the turning of some small steel articles, and we have no doubt that there are other instances, especially where we are prepared to build stronger machines, where similar and even greater savings can be effected."

One phase of milling, in which cutters made from high-speed steel will play an even more important part than at the present time, will be that of gear- and similar cutting, it now being possible to make formed backed-off cutters of high-speed steel for all purposes, and the results obtained from the use of them are shown in the appendix, page —, prove beyond all doubt that very great advantage is to be gained from their application and use.

Again, rapid cutting with planing-tools has also developed extensively, the old cutting-speeds of from 15 to 25 ft. per min. being now replaced by those of from 50 to 60 ft., and in some cases even as high as 80 ft. per min., and for the same reasons, as already described in lathe-turning, the power absorbed does not increase in anything like the same proportion to the extra amount of work done, so that the wear and tear on the machine is not materially increased.

It was for some time not thought possible to plane at such

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high speeds on account of the tools coming into contact suddenly with the job, and running risks of snapping-off through shock, but where high-speed steel of proper quality is used, this difficulty is overcome, and a good example or two of rapid planing may be quoted. Using a 7-ft. planing-machine with two tools operating on forged steel of medium quality, the cutting-speed, depth of cut and feed of each tool is respectively 54 ft., 0.25 in. and $\frac{1}{8}$ in., the speed of reverse being 160 ft. per minute.

Another striking example of high-speed planing on a large cast-iron turbine body was:—Cutting-speed 36 ft. per min., depth of cut 1.25 in., and feed seven cuts per inch, the tool cutting for ten hours without necessitating grinding. Two tools were cutting, each taking a cut as described, the size of the planer being 14 ft. by 30 ft. An illustration of these machines at work is shown on Figs. 14, 15 and 16.

The question of cutting-angles for tools is an important one, and the author would advise all interested to peruse the paper written by Professor Nicolson of Manchester, and read before the Institution of Mechanical Engineers at Chicago this year, and in which he states that the best cutting-angle as deduced from the results of experiments is 75° for steel and 80° for cast-iron. Of course, these angles may with advantage be modified according to circumstances and the nature of any particular class of work.

Objections have been made against high-speed steel on the grounds of its being brittle, but this is not the case where the steel has been properly annealed and the hardening confined to the cutting area, and sufficient support given to the tools when fixed in the machine.

An example of the great pressure-resisting powers of high-speed steel may be given.

When cutting forged steel of about 30 tons per sq. in. tensile-strength and offering a resistance to cutting of about 100 tons per sq. in., a tool of 1.25 sq. in. section was used, taking a cut of $\frac{7}{8}$ in. in depth by 0.25 in. feed per rev., equivalent to an area of metal under cut of 0.21875 sq. in., the cutting-speed being 90 ft. per min., and removing 68.75 lb. of metal per min., or the huge weight of 4,010 lb. per hour. The tool in this instance was projecting a distance of $1\frac{1}{8}$ in. beyond the

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rest (see Fig. 17), and a calculation shows the stress on the tool to be as high as 78.5 tons per sq. inch. In another case, cutting forged steel of 35 tons tensile-strength and offering a resistance to cutting of 115 tons per sq. in., a 1.25 in. square tool being used, the diameter of forging was reduced by 1 in., equal to 0.5-in. depth of cut, whilst the tool advanced $\frac{3}{8}$ in. every revolution, the cutting-speed being 25 ft. per min. and removing 14.25 lb. of metal per minute. With the point of the tool projecting 0.75 in. beyond the rest, the tool successfully withstood a stress of 51.6 tons per sq. in. (see Fig. 18).

Although in actual practice, tools of much greater section would be used, the results clearly show that, if proper care be taken, tools of high-speed steel are quite capable of withstanding any pressure likely to be met in ordinary workshop-practice.

A most important point to observe when taking heavy cuts is that of having the tools quite flat on the bottom side and supported as near as possible up to the extreme edge, as by so doing the pressures tending to break the tool are very considerably reduced. For example, the position of the tool as placed in the rest shown in Fig. 18, would cause a stress of something like 78.5 tons per sq. in. to be thrown on it, whereas, when the overhang is reduced to one-half of the original distance, equal to $\frac{9}{16}$ in., the stress is lowered to 14.27 tons per sq. in., a reduction of 80 per cent.

Perhaps one of the most unlooked-for developments in the use of high-speed steel has been the manufacture from it of twist-drills, and it would be safe to say that in no other sphere has the new steel justified itself to a greater extent than in the operations of drilling and boring, as its powers in that respect have revolutionized completely modern workshop-practice. It is now possible in many cases to drill holes through stacks of thin steel plates as quickly and economically as by punching them, thus avoiding the consequent liability to distress the material due to the punching-action.

The plates of torpedo and other boats, which are comparatively thin and of high tensile-strength, can now be drilled in stacks with such facility that it is no longer necessary to punch the holes, whilst in many articles where it was formerly the practice to core in the holes, as, for example, in cylinder and other covers, or pipe flanges, etc., it is now cheaper and quicker

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to use high-speed steel and drill the holes out of the solid. Many examples might be adduced in support of these statements, but reference is made to a few striking instances, full particulars of which are given in the appendix.

A short time back the author received a letter from a large firm of structural engineers in Glasgow giving some remarkable drilling results with high-speed drills, and the following extract may be quoted:

"Drilling mild steel 2.5 in. thick made up of five $\frac{8}{8}$-in. plates and one $\frac{5}{8}$-in. angle iron, a $\frac{15}{16}$-in. diameter twist-drill made of high-speed steel, and running at 275 rev. per min., with a feed of 75 cuts per in. of penetration, drilled 7,924 holes without requiring regrinding, each hole being drilled in 42 seconds."

As a comparison of the superiority of high-speed over ordinary drills, an instructive result was obtained when drilling forged steel gun-cradles of 5-in. thickness, which steel is of a very tough nature. An ordinary twist-drill was first tried and failed after drilling 8 holes, the end being completely fused, but a high-speed drill afterwards drilled 124 holes without suffering any injury whatever. The drills were 2 in. in diameter, running at 80 rev. per min., and each hole was drilled in six minutes, this being the full power of the machine.

A further illustration of the economy resulting from the use of high-speed twist-drills may be gathered from the fact that the author's firm in several instances reduced the cost of drilling per 100 holes by over 60 per cent. without even altering the machines in any way, except by speeding them up.

On cast-iron equally good results are obtained. Opening out cored holes in cast-iron girders, from $\frac{5}{8}$ in. to $\frac{15}{16}$ in. in diameter, the high-speed drills do four and a half times as much work as the drills previously used, in addition to lasting considerably longer without grinding, whilst in another operation where the holes are drilled out of the solid the difference in production is eight times in favor of the high-speed drill.

Equally satisfactory results are obtained from flat drills and other form of boring-tools made of high-speed steel for use in boring large holes out of solid bars, shafts, forgings, etc., the author's firm daily using drills of this type up to 12 in. in diameter and boring holes up to 50 ft. long.

Other miscellaneous uses to which high-speed steel may be

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applied include taps, screwing-dies, reamers, hot punches, circular saws, both solid and with teeth inserted in mild-steel bodies, marble-drilling, marble-planing, rock-drills, etc., also in the case of articles whose surfaces are subject to hard wear, such as lathe-centers, tube-expanding tools, etc.

Finishing.—A considerable amount of doubt has been thrown from time to time on the inability to take finishing-cuts with high-speed steel, and in the early stages of its development this contention was to a large extent justified, but experience and practice have brought the steel into line and rendered it possible to obtain an excellent finish at high speeds with tools suitably formed and properly arranged in the machines. Some very good examples of finished bright work at high speeds are exhibited on the table, which have been mostly made in semi-automatic machines, high-speed steel being used and one cut only taken. It will be seen on examination that the surface-finish is most excellent.

These samples, along with many others placed on the table, also illustrate the finishing powers of high-speed steel when used in machines suitably adapted and they have been turned from the rolled bar with one cut only, being guaranteed accurate in diameter to 0.002 in., whilst the excellence of surface-finish will compare with the best obtainable by the old system of cutting, finishing and polishing.

This finishing-quality of high-speed steel is especially advantageous for tools used in automatic and capstan lathes, because it enables the work to be produced so very much more rapidly, and also on account of the great resistance of the steel to wearing action greater accuracy is ensured.

As regards the quality of retaining a sharp edge, high-speed steel makes excellent razors, and will long retain without sharpening an extremely keen cutting-edge. The author may add that it is thus now possible to those whose time is precious to indulge even in "high-speed shaving."

The author hopes that the few facts he has given as to the use and development of high-speed steel may indicate some of its uses and progress, but he can scarcely refrain from remarking that many are saying, and rightly so, "Yes! these results are very remarkable; but what of the machines to perform such prodigious work?" and this leads him to speak, before

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concluding, as to how one important development often leads up to another of even greater magnitude, and that is in this case the complete revolution in the design of machine-tools to cope with the extraordinary increased cutting-powers of the latest rapid cutting-steels.

It is impossible that the design of machine-tools can remain on the old lines, since the difference between them and the cutting-powers of the steel is so abnormal, and a sphere of immense area for the redesigning of machine-tools is opened out to the ingenuity of the world's engineers.

That much has been already done is admitted, but the work is naturally of such a nature that only time and experience will accomplish, gradually enabling as nearly as possible the relative powers of the steel and machines to be equated.

In the machine-tool department of the author's firm, this branch of the subject of remodeling their tools has received the closest attention, and a type of their modern 18-in.-center lathe for high-speed cutting may be mentioned. It is capable of exerting 65 h.p. equivalent to a belt width of 12 in., and with the aid of a variable-speed motor a range of cutting-speeds from 16 to 400 ft. per min. are possible, this comparing with an old type 18-in. lathe having a belt of 4-in. width, and capable of exerting only about 12 horse power.

In a similar way the old types of planing-, milling-, drilling-machines, etc., are all more or less obsolete, and new designs are already constructed to cope with work at speeds and feeds described in this paper.

It is indeed a pleasure to see the new type of machine-tool operating with high-speed steel, and treating the work it has to turn out in such a business-like way, throwing off shavings from steel and iron as one usually sees in turning wood, and imparting a life and energy to the whole establishment in remarkable contrast to the sleepy rate at which metals used to be turned and machined for so many years past, thus exerting an influence on everybody therein to get "a hustle on" that is positively exhilarating in its effects.

The chemists and metallurgists, with the skill they have brought to bear in assimilating the metals provided by nature to make so wonderful a product as this high-speed steel, have set a formidable task to the mechanical world, but it is satisfac-

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tory to observe that in most quarters, engineers, although for the time being in a secondary position, are roused to exercising their fullest energies in getting level. This competition can only be considered as a healthy sign, since it will give further impetus to the steel-maker, and remind him to continue to fathom more and more the mysteries and powers of this steel.

In concluding this paper, the author would add it cannot be denied that the question of high-speed steel is one of vast importance to all manufacturing-countries, for if they are to hold their position in the world's competition they will be compelled to study the use and advantages to be derived from its adoption, and, in addition, to bring up their machinery into the most modern and efficient form for obtaining the utmost producing capacities.

The powers of the cutting-steel and the machine should be as nearly reciprocal as possible, for, given those conditions, manufacturers place themselves in the most favorable position for the rapid and economical turn-out of their products, and for best meeting their rivals in the open field of competition; and, lastly, notwithstanding the wonderful results obtained with the new cutting-steel, and developed comparatively so rapidly, nothing approaching finality must be admitted by the steel-maker, but incessant diligence, experiment and research to discover, still more, the best combinations of nature's metals, should be his constant aim, and in the doing of which not only is there much pleasure derivable to himself, but in addition there is the benefit it is possible to render to the whole world.

Appendix.

The following results with high-speed steel, the analysis of which is given on p. 342 of this paper, will be of interest. (See Figs. 11 to 16.)

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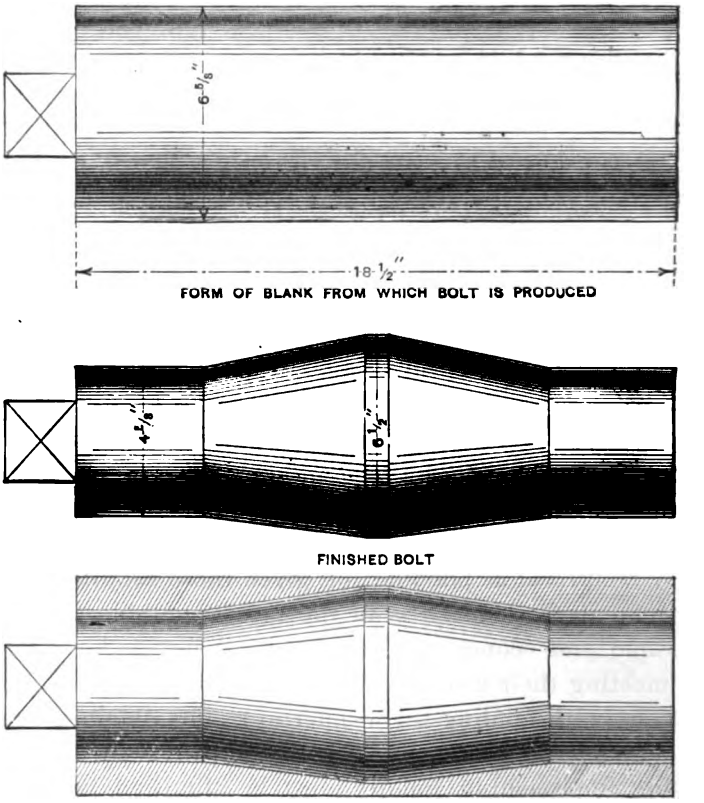
FIG. 19.



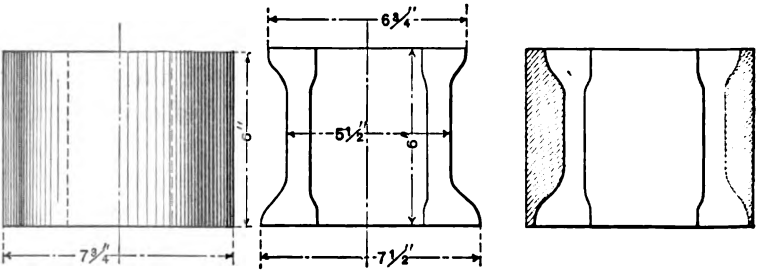
FORM OF BLANK FROM WHICH BOLT IS PRODUCED

FINISHED BOLT

Cutting-speed, 160 ft. per minute. Mean depth of cut, $\frac{3}{4}$ inch. Traverse, $\frac{1}{12}$ inch. Reduction in weight, 62 pounds. 40 bolts turned in one day with "A. W." high-speed tool-steel. Sectioned area indicates the metal removed. Total weight of metal removed in 10 hr., 2,480 pounds.

ARMOR BOLTS.

FIG. 20.



Form of finished sleeve. Blank from which sleeve is produced.

Cutting-speed, 160 ft. per. minute. Maximum depth cut, $1\frac{1}{8}$ inch. Traverse, $\frac{1}{12}$ inch. Reduction in weight, 36 pounds. 60 sleeves produced per day of 10 hr. in 12-in. high-speed lathe using "A. W." steel. Sectional area indicates the metal removed from each blank. Total weight of metal removed in 10 hr., 2,160 pounds.

SLEEVES FOR ARMOR-PLATE BOLTS.

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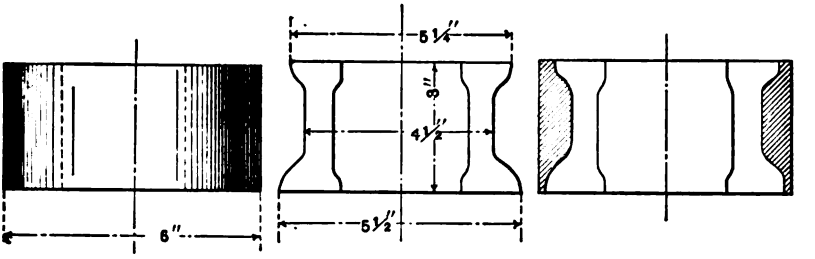
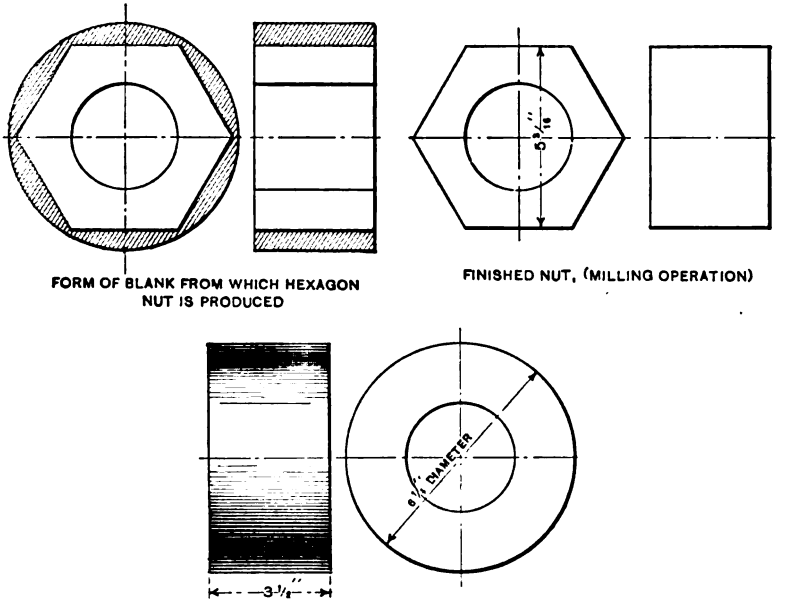
FIG. 21.



Cutting-speed, 150 ft. per miuute. Maximum depth of cut, $\frac{3}{4}$ inch. Reduction in weight, 8.37 pounds. 90 sleeves produced per day of 10 hr. with "A. W." high-speed milling-cutter. Sectioned area indicates the metal removed from each blank. Total weight of metal removed in 10 hr., 754 pounds.

SLEEVES FOR ARMOR-PLATE BOLTS.

FIG. 22.



Speed of cutter, 150 ft. per minute. Maximum depth of cut, $\frac{1}{32}$ inch. Reduction in weight, 7.5 pounds. 90 sleeves produced per day of 10 hr. with milling-cutter made of "A. W." steel. Sectioned area indicates metal removed. Total weight removed in 10 hr., 675 pounds. (Sir W. G. Armstrong, Whitworth & Co., Ltd., Manchester.)

HEXAGON NUTS FOR ARMOR-PLATE BOLTS.

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A. Tests With High-Speed Seeel.

The remaining results of the latest workshop-practice have been kindly supplied by various engineering firms in Great Britain; and will give some idea of the great advancement in cutting-speeds since the introduction of the steel.

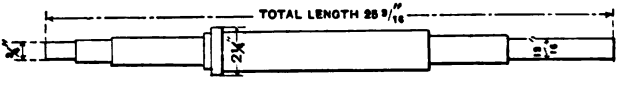
Turning.

"The samples referred to at foot have been produced on a No. 3B hexagon turret-lathe, and made from the bar direct, using high-speed steel, and upon the labels attached to the samples, details of the cutting-speed, feed, etc., are given, also actual time required for machining each sample.

"These samples were made, not to illustrate what is the largest amount of metal which can be removed in a given time (this regardless of finish), but what may be considered to be a fair time for producing a good job, and all the reductions on the samples were taken at one cut; no finishing cut is taken, and no filing or polishing is done."

Figs. 23 to 28 inclusive illustrate finished work produced at high-cutting speeds.

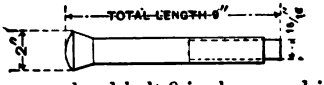
Fig. 23.



Sample No. 1.—This is 25¼ in. long and was made from a bar slightly larger in diameter than the collar; it has seven different diameters and a screw thread, and was made in 45 min., and the reduction, as will be noted, is fairly large.

Time, 45 minutes. Cutting-speed—maximum, 37.6; minimum, 12.5 ft. per minute. Feed, 92 per in.; traverse, 1 in. in 1 min. 26 seconds.

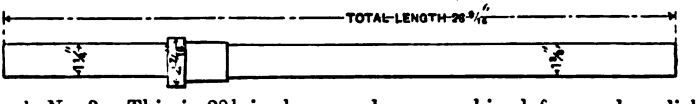
Fig. 24.



Sample No. 2.—This is a cone head bolt 9 in. long, and is a very good example of work indeed, and was machined from the bar in 13 minutes.

Time, 13 minutes. Cutting-speed—maximum, 58.5; minimum, 27.4 ft. per minute. Feed, 92 per in.; traverse, 1 in. in 49 seconds.

Fig. 25.



Sample No. 3.—This is 28½ in. long, and was machined from a bar slightly above the diameter of the collar in 37½ minutes.

Time, 37½ minutes. Cutting-speed—maximum, 49.5; minimum, 25.4 ft. per minute. Feed, 92 per in.; traverse, 1 in. in 1 min. 11 seconds.

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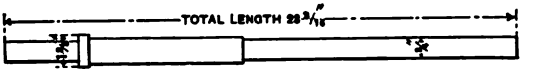
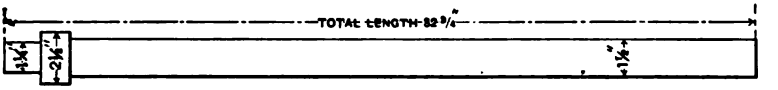
FIG. 26.



Sample No. 4.—This is 23¼ in. long, was made from a bar ¼ in. larger in diameter than the collar; this is rather a slender job, and was machined in 21 minutes.

Time, 21 minutes. Cutting-speed—maximum, 50.4; minimum, 27.4 ft. per minute. Feed, 92 per in.; traverse, 1 in. in 39 seconds.

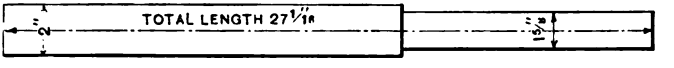
FIG. 27.



Sample No. 5.—This is 32¾ in. long, was made from a bar ⅛ in. larger in diameter than the collar, and was made in 33½ minutes.

Time, 33½ minutes. Cutting-speed—maximum, 41.8; minimum, 20.9 ft. per minute. Feed, 62 per in.; traverse, 1 in. in 58 seconds.

FIG. 28.

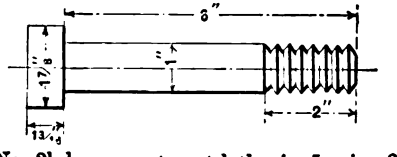


Sample No. 6.—This is 27 in. long, was turned from a bar ¼ in. larger than the largest diameter, and was machined in 23 min. 40 seconds.

Time, 23 min. 40 seconds. Cutting-speed—maximum, 58.5; minimum, 47.6 ft. per minute. Feed, 92 per in.; traverse, 1 in. in 49 seconds.

Illustrations of rough bolts made on a No. 2b hexagon turret-lathe at high-cutting speeds are given in Figs. 29 and 30.

FIG. 29.



Produced upon a No. 2b hexagon turret-lathe in 5 min. 28 sec., using high-speed tools. The following are the details:—

Reducing 1¼-in. bar to 1 in., 6 in. long, 96 cuts per in.,	4 minutes 0 seconds.
Screwing,	40 "
Cutting-off,	28 "
Idle movements, etc.,	20 "
Total,	5 minutes 28 seconds.

The above time does not include facing the back of the head, which would require another half minute.

This bolt, made in the ordinary way with ordinary tool-steel, with a good operator, requires 15 minutes.

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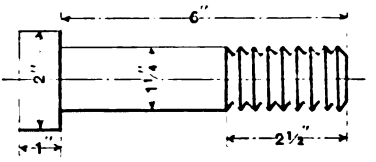
FIG. 30.



This sample was made with a special tool-holder, using two high-speed tools, one at the front and one at the back, and was intended to determine what could be done in the way of producing a rough bolt quickly. The total time was 2 min. 18 sec., made up as follows :—

Turning-speed, 144 revolutions; feed, 33 per in.,	1 minute 25 seconds.
Screwing,	23 "
Cutting-off,	30 "
Total,	2 minutes 18 seconds.

General Shop Work.

"Much of the work dealt with, especially the cast-iron work, is quite light and will not stand very heavy cuts, and up to the present it has not been possible to instal special machinery for heavy cutting with high-speed steel to any great extent. It will be seen, however, that the figures given represent a very good duty on the part of the cutting-steel."

Rough Turning Mild Steel Bars with high-speed steel on No. 5 hexagon turret-lathe. Bars, 4.5 in. diameter; turning, 31.5 ft. per min.; depth of cut, 0.25 in.; feed, 0.25 inch. The above is the result of a test. In actual practice higher cutting-speeds and finer feeds are used, as the bar is not supported and will not stand these very heavy cuts when projecting very far from the chuck. The tool was lubricated with a liberal supply of lard-oil.

Rough Turning Capstan Lathe Spindles with high-speed steel on 12-inch lathe. Cutting-speed, 58 ft. per min.; depth of cut, $\frac{1}{4}$ in.; feed, 16 cuts per inch.

The above spindles are made of steel containing 0.5 per cent. of carbon. The tool was lubricated with a liberal supply of suds. The limit of the work in this case was the lathe and not the tool. At another test the same work was done at 63 ft. per minute. Cut, $\frac{3}{8}$ in. deep; feed, 32 cuts per inch.

Boring Lathe Spindles out of the solid bar with high-speed flat cutter. The material from which these spindles are made contains 0.5 per cent. of carbon. Diameter of holes, $3\frac{9}{16}$ inches. Bored out of the solid at 64 turns per min. = 60 ft. per minute. Feed, 85 cuts per in., 45 in. travel per hour.

Pulley Turning with high-speed steel-tools. Roughing out, 60 ft. per min.; cut, $\frac{5}{32}$ in. deep; feed, $\frac{3}{32}$ in. per rev.; finishing cut, 100 ft. per min.; cut, $\frac{1}{32}$ in. deep; feed, 36 cuts per inch.

The pulleys in question are very thin, and will not stand a heavy cut.

Milling.

Milling Medium Cast-Iron with face-milling cutter 4.5 in. in diameter, with 14 inserted teeth made of high-speed steel, cutting at 70.7 ft. per minute. Depth of cut, $\frac{5}{32}$ in.; width of cut, 3 in.; feed, 8 in. per minute.

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Milling Medium Cast-Iron with face-milling cutter 9 in. in diameter, 12 inserted teeth made of high-speed steel, cutting at 80 ft. per minute. Depth of cut, $\frac{1}{8}$ in.; width of cut, 6 in.; feed, 8.5 in. per minute.

Milling Medium Cast-Iron with face-milling cutter 12 in. in diameter, with 16 inserted teeth made of high-speed steel, cutting 75 ft. per minute. Depth of cut, $\frac{5}{32}$ in.; width, 11 in.; feed, 7.5 in. per minute.

Cylindrical Cutter Milling Mild Steel.—Cutter made from high-speed steel, 2.75 in. in diameter, 21 teeth, running at 72 ft. per minute. Depth of cut, $\frac{5}{32}$ in.; width of cut, 3 in.; feed, 7.5 in. per minute.

Cylindrical Cutter Milling Mild Steel.—Cutter, 3 in. in diameter, 10 teeth, running at 70 ft. per minute. Depth of cut, $\frac{5}{32}$ in.; width, 11 in.; feed, 3 in. per minute.

Cylindrical Cutter Milling Cast-Iron.—Cutter, 2.5 in in. diameter, 21 teeth, running at 87 ft. per minute. Depth of cut, $\frac{1}{16}$ in.; width, 2.75 in.; feed, 26 in. per min. = 0.195 per turn = 0.0093 per tooth. With same cutter, finishing-cut was taken, 155 turns per min. = 111 ft. ; depth of cut, $\frac{1}{64}$ in.; feed, 6.5 in. per minute.

Gear-Cutting, 5 in. pitch. Stocking-cutter, 3.5 in. in diameter, cutting cast-iron spur-gear from solid. Cutting-speed, 75 ft. per min.; feed, 14 in. per minute.

At the time the observation was made 5,000 teeth, 2.25 in. long, had been cut. The cutter appeared to be in as good condition as after cutting the first 100 teeth (thirteen and a half hours, nearly).

B. Turning.

High-speed steel has so far been very successful on roughing 0.34 per cent. carbon-steel armature-shafts; the average time the tools last is about 4 hours. Occasionally a tool will last one day; this is owing to the variation in the steel being cut.

1. The cutting-speed on this work is: 54 ft. per min.; 11 rev. per in. of feed; $\frac{1}{8}$ to $\frac{5}{16}$ in. depth of cut. The operator grinds his tool two or three times a day.

2. A high-speed steel-tool, 1.25 by 2.5 in. in section; depth of cut, 1.25 in.; feed, 7 cuts per in.; cutting-speed, 36 ft. per minute. Planed across the base of a 5,500 k.w. steam-turbine body; the tool worked for 10 hours before requiring grinding.

3. Tool, 2 in. square high-speed steel; used in 28 ft. boring-mill; cutting cast-iron ring 16 ft. 6 in. in diameter; cutting-speed, 40 ft. per min.; depth of cut, 1 in.; feed, 5 3 cuts per in.; tool worked 15 hours.

4. Machine used, hexagon turret-lathe; tool, roughing, 0.5 by 1 in. in section; operation, reducing a 1.75 in. rolled-steel bar to 0.75 in. diameter; cutting-speed, 38 ft. per min.; feed, 100 rev. per 1 in. of feed. The tool lasted for over 46 hr. before regrinding was required.

5. Operation, reducing cold rolled-steel bars by $\frac{3}{32}$ in. in diameter; cutting-speed, 160 ft. per min., feed, 3.75 in. travel per minute. The tool lasted for 23 hr. before requiring regrinding.

Milling.

6. "Some phenomenal results have been obtained with milling-cutters on special tests; of course the work was not good enough for practical use, being furrowed like a ploughed field, but the cut was clean enough, the object being to get at the amount the cutters would stand before breaking the corners."

Slot cutter, $3\frac{1}{4}$ in. in diameter, $\frac{5}{8}$ in. wide, 1 in. bore; 180 rev. per min. = 147 ft.;

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0.050 in. per rev. = 9 in. travel per min.; 0.25 in. depth of cut; material cut, mild steel. Cut for 1 hr. in perfect condition without grinding.

With cut ¼ in. deep, the feed was put up gradually until 42 in. table travel per min. was obtained, at which feed the machine pulled up; 37.75 in. per min. was run with success. It was not considered advisable to break the corners of this cutter, after behaving so well, the cutter running for about 5 hr. on the severest test. Although well flushed with water, the chips came off blue, and were like planer chips when broad cutting. The cutter showed very little sign of wear.

7. Slot cutter, 3½ in. in diameter, 1 5/16 in. wide, 1 in. bore; 120 rev. per min. = 98 ft.; 0.168 in. feed per min. = 19.75 in travel; 1/10 in. depth of cut.

In cutting cast-iron, this cutter ran over 20 sq. ft. of surface, and showed very little sign of wear when taken out.

Milling and Gear-Cutting.

8. Straddle-cutter, 3 in. in diameter, 1 in. bore; cutting-speed, 70 ft per min.; feed, 0.20 in. per rev., 1¼ in. travel; depth of cut, ¾ in. (in one cut).

Cutting slot, 0.75 in. square, in end of mild-steel blocks, the cutter did 2,200 pieces before requiring regrinding, lasting 80 hours.

9. Tool, steel angle cutter, 50–12; operation, milling tool-steel; cutting-speed, 72 ft. per min.; feed, 0.03 in., or 2.7 in. travel per min., depth of cut, 5/16 inch.

The cutter worked for 3 hr., and was then in such good condition that the speed was put up to 84 ft. per min., and the feed to 0.036 in., or 3.8 in. travel per minute The cutter then stood for 8.5 hr. before it required regrinding.

Following the previous tests, trials were made with high-speed steel for cutting gears, with results as follows:—

10. *High-Speed Cutter*, 8 in. in diameter, 2.5 in. pitch; cutting-speed, 57 ft. per min.; feed, 3⅛ in. per min.; depth of cut, 0.862 inch.

Cutter lasted for three days before requiring regrinding.

Ordinary Carbon-Steel Cutter, same size as above, as follows: Cutting-speed, 27 ft. per min.; feed, 1⅛ in. per min.; depth of cut, 0.862 inch.

Average life of above cutter, made of ordinary carbon-steel, is about 3.5 hours. These cutters were used on cast-steel street-car motor-gears, 0.50 per cent. of carbon.

Gear-Cutting and Drilling.

Cutting pinions, 0.34 per cent. carbon-steel, for street-car motors.

11. *High-Speed Cutter*, 7 in. in diameter, 2.5 in. pitch; 57 ft. per min.; 2⅜ in. feed per min.; depth of cut, 0.862 inch.

Average time cutter lasted, without grinding, was about 27 hours.

Ordinary Carbon-Steel Cutter, 7 in. in diameter, 2.5 in. pitch; 30 ft. cutting-speed per min.; 1 1/16 in. feed per min.; depth of cut, 0.862 inch.

Average time cutters lasted, without grinding, was about 7 hours.

Tests with High-Speed Twist-Drills.

12. Tool, 0.75 in. drill; operation, drilling cast-iron, medium hard, 4 in. thick; 360 rev. per min.; feed, 60 rev. per in. of feed = 6 in. per min.; 137 holes were drilled before the drill required regrinding.

13. Tool, 1-in. drill; operation, drilling cast-iron, medium hard; 240 rev. per min.; feed, 5 rev. per in. of feed = 4⅜ in. per min.; 78 holes were drilled before the drill required regrinding.

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C. General Shop Work.

Particulars of Work Done by High-Speed Tool-Steel.

Operation.	Depth of Cut.	Feed.	Cutting Speed per Minute.	Material.
	Inches.	Inches.	Feet.	
Turning.....	$\frac{1}{2}$	$\frac{1}{16}$	40	Steel crank axles, roughing.
	$\frac{3}{8}$	$\frac{1}{8}$	50 to 70	Shaft for built-up cranks, roughing.
	$\frac{1}{16}$	$\frac{1}{16}$	120 to 140	Shafts for built-up cranks.
	$\frac{1}{4}$	$\frac{1}{16}$	60 to 70	Steel piston-rods, roughing. Crosshead gudgeon pins, bolts for crank pins, etc., C. I. piston heads.
Vertical boring-machine.	$\frac{1}{8}$	$\frac{1}{16}$	50	C. I. piston rings, boring, turning and parting off.
Inserted tools in large milling-head.	$\frac{1}{8}$	$\frac{1}{8}$	55	Pins of steel cranks. *Note.*—Tools are subject to impact at each revolution of the wheel. Number of tools in head, 48.
Planing.....	$\frac{1}{2}$		25	Steel slabs for built-up cranks.
	$\frac{1}{2}$		35	Steel horn blocks, etc., also on cast-iron.

NOTE.—The above are regular speeds and feeds, and involve about the same amount of grinding as with the Mushet steel at lower speeds. In the case of crank-pin milling, speed has been increased from 20 to 55 ft. per minute.

D. Tests with High-Speed Steel.

General Shop Work.

"The following are a few rough notes relating to the performance of high-speed steel:"

1. When turning ends for large rolls with a cutting-speed of 80 ft. per min. on hard steel; depth of cut, $\frac{1}{8}$ in., and a feed of 8 to the inch. In this case the tool was cutting 7 hr. before being reground.

2. When turning piston-rods with a cutting-speed of from 110 to 120 ft. per min.; depth of cut, $\frac{3}{16}$ in.; feed, 16 to the inch. In this case a heavier cut could have been taken.

3. Turning cones cut from 6-in. bore solid drawn steel tubes with a cutting-speed of from 160 to 180 ft. per minute. By using high-speed steel the output was rather more than doubled.

4. On ordinary cast-iron a speed of 80 ft. per min. was attained with cuts to suit the class of work in hand, but no difficulty is found with cuts 0.5 in. deep, $\frac{1}{8}$ in. feed.

5. Since adopting high-speed steel on planing-machines, steel castings have been cut at 30 ft. per min. instead of 17 ft. as formerly.

6. High-speed steel for milling-cutters has given every satisfaction. On all ordinary light work from 160 to 240 ft. per min. have been cut with very satisfactory results.

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E. Test with High-Speed Steel and Cast-Iron.

"Enclosed is a report relative to testing of high-speed steel."

Test on forged crank-shaft, 22 in. in diameter, was as follows:—Cutting-speed, 63 ft. per min., 0.25 in. feed; 0.25 in. deep at this speed and feed, the tool without being reground cut for 4.5 hr.; at the end of this time the tool was not punished.

Cast-Iron.—Turning fly-wheel, 16 ft. in diameter; cutting-speed, 50 ft. per min.; $\frac{1}{8}$ in. feed; depth of cut, 0.75 inch.

For cast-iron, such as fly-wheels and rope-pulleys, cylinder-covers, etc., the cutting-speed on an average is from 50 to 60 ft. per min.; the depth of cut and feed varying from $\frac{1}{16}$ to $\frac{1}{4}$ in. per revolution.

F. Tests with High-Speed Steel.

General Shop Work.

"The speeds indicated below are those of regular practice, but at the same time it may be stated that even better results than these are possible, on small shafting, cutting-speeds of 200 ft. per min. being easily attainable."

Rough turning with high-speed steel.

Hard Cast-Iron Rolls.—50 ft. per min., $\frac{1}{4}$ in. feed, $\frac{5}{16}$ in. deep. Roll, about 20 in. in diameter by 60 in. long. A tool will turn three or four rolls without grinding.

Cast-Steel.—30 to 40 ft. per min.; feed varying according to cut.

Mild Steel.—90 ft., 0.25 feed, 0.25 in. deep.

Planing.—Cast-iron and steel, 36 ft. per minute.

Drilling.—4 in. depth per min. up to 1.25 in. in diameter, cast-iron, wrought-iron, or cast-steel.

G. Test with Cast-Iron Drum.

"The following is a very good example of rapid cutting on a cast-iron drum, 12 ft. in diameter by 5 ft. face; this was turned from the black at a speed of about 35 ft. per minute. The second cut was made at a speed of 50 ft. per min.; depth of cut $\frac{1}{8}$ in., with $\frac{1}{12}$ in. traverse. In this instance the tool completed the entire surface of the drum without once regrinding, and was in excellent condition at the termination of the work. Two similar cuts have also been done over a drum of exactly the same diameter as above with the same tool, without regrinding from start to finish, at a speed of 50 ft. per min. and without losing $\frac{1}{64}$ in. in girth on the drum during the operation (27.25 hr. cutting)."

H. Tests with High-Speed Tools.

"A few examples of the results of work with high-speed steel and drills are herewith submitted, the same being results of daily usage and not test results. Much higher average results could be obtained were the belt powers of the lathes and drilling-machines not rather limited."

Turning.—Roughing mild steel bars of $3\frac{3}{8}$ in. diameter for worm shafts. Maximum speed, 150 ft. per min.; minimum speed, 70 ft. per min.; depth of cut, $\frac{1}{16}$ in.; feed, 5 in. travel per min.; tool works from 7 to 8 hr. without grinding.

Facing Cast-Iron.—Speed, 75 ft. per min.; depth of cut, $\frac{1}{8}$ in.; feed, $\frac{1}{16}$ inch.

Turning Hard Cast-Iron Ram.—Speed, 45 ft. per min.; depth of the cut, $\frac{5}{16}$ in.; feed, $\frac{1}{16}$ inch.

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Planing.—Speed, 34 ft. per min. ; depth of cut, from $\frac{3}{8}$ to $\frac{1}{2}$ in. ; feed, $\frac{1}{12}$ in. ; tool works 6 hr. without grinding.

Drilling with High-Speed Twist-Drills.—0.75 in. drill, drilling cast-iron and mild steel ; speed, 200 rev. per min. ; feed, $1\frac{1}{8}$ in. per minute.

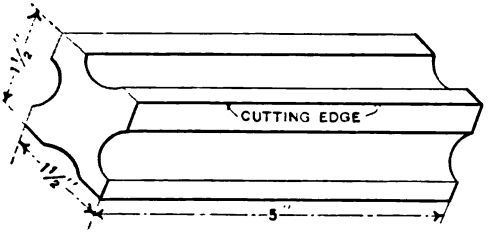
Ordinary Twist Drills will not stand at the speed at which high-speed drills are run, and are destroyed immediately, the practice being to run ordinary drills two speeds slower than high-speed drills.

I. Tests with High-Speed Steel.

Drilling.

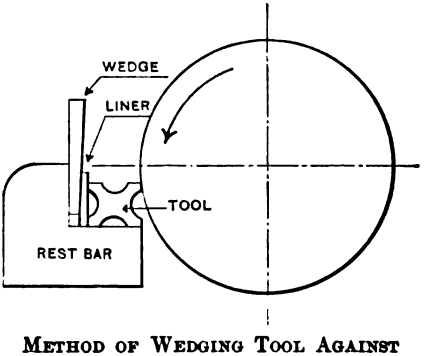
"A high-speed twist-drill $1\frac{1}{16}$ in. in diameter used for drilling steel girders consisting of five $\frac{3}{8}$-in. plates and one $\frac{5}{8}$ in. angle iron = 2.5 in. thick in middle of girders and 1 in. at ends, drilled 7,924 holes before sharpening, and in the case of another similar drill 6,504 holes were drilled before sharpening, the drills running 275 rev. per min. with a feed of 75 cuts per in., equal to a feed of 3.6 in. per minute. Formerly when using ordinary twist-drills of 0.75 in. and $\frac{7}{8}$ in. in diameter the speed ranged from 96 to 115 rev. per min., and feeding about $\frac{5}{8}$ to 1 in. per minute. These figures speak for themselves, the life of the ordinary drills formerly used being well known to most engineering firms."

Fig. 31.



Shape of Cutting-Tool.

Fig. 32.



Method of Wedging Tool Against the Barrel and Roll.

Roll Turning.

"As is generally known, the surfaces of chilled cast-iron rolls are so extremely hard that, in turning them, only the very best steel is of any use, and the cutting-speed must be exceedingly slow.

Moreover, in turning large plain chilled rolls, sharp or narrow-pointed tools, such as are used on general traversing lathe work, would not stand at all, the form found to answer best being as shown in Fig 31. These tools are wedged against the barrel of the rolls as shown in Fig. 32.

Hitherto the circumferential cutting-speed generally adopted has been about 6 ft. per min., and the reduction of the metal very small, as will be seen by examining the sample box of turnings marked (A), which were cut off at the speed referred to with a tool of the best usual quality of steel. With this tool, if the

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cutting-speed was increased, or a heavier cut put on, the edge of the tool would promptly fail. By using high-speed steel it was found practicable:

1. To increase the cutting-speed from 6 ft. per min. to 11 ft. per minute.

2. To put on a much heavier cut, as will be seen by comparing the sample turnings (B) with the lighter sample (A), cut with the best usual steel.

3. In addition to the above advantages of quicker work and heavier cutting, the special tool referred to will last much longer without taking out for self-cooling or grinding."

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SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A Special Form of Slag-Car.

BY L. J. W. JONES AND B. H. BENNETTS, TACOMA, WASH.

(Annual Meeting, February, 1905).

THE removal and disposition of large quantities of slag from blast-furnaces is a question of great importance in the design of works, and various methods have been devised, from time to time, in order to take the best advantage of local conditions.

In blast-furnaces treating copper-ores or lead-ores, it is necessary to use a fore-hearth, or matte-settler, in order to collect, if possible, every particle of valuable matte which, otherwise, would be carried away with the out-flowing slag. In all of these cases, the slag overflowing from the fore-hearth is caught in pots and then conveyed to the dump. If the slag is of no value and the necessary water-supply, grade and dumping-space be available, the disposal of the slag by granulation is by far the cheapest and best method. It involves a minimum of labor, both at the furnace and at the dump; and it permits a continuous outflow of slag from the furnace- or settling-pot, thereby avoiding frequent tapping and plugging. When the conditions of water-supply, grade and dump-area will not permit slag-granulation, there are three other methods in common use. Large steel or cast-iron pots, or ladles, supported by trunnions on heavy cars which are moved by steam-, air-, or electric locomotives; some type of mechanical conveyor; and small pots or buggies moved by hand. The pots on the slag-cars are generally circular or oval in shape, and each one contains from 15 to 40 or more cu. ft. of molten material. The mechanism for tipping them is more or less elaborate; and, as a general rule, the slag is dumped alongside, and quite close to, the track. They are more or less costly to construct and to keep in repair, and, in order to use them to best advantage, it is necessary to have a good elevation of track above the dumping-ground; furthermore, the rails must be of heavy weight, and well-supported on heavy ties. Other disadvantages of the slag-pots are—1, that the slag cannot be removed continuously

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from the furnace, which must therefore be frequently tapped and plugged; 2, that the resultant cone-shaped masses of slag are unwieldy and difficult to handle; 3, that the slag is dumped at the side of the car only, and 4, that the pots cannot be brought close to the slag-spouts. For small plants the method of slag removal by large pots is too expensive, small pots moved by hand, or larger pots of from 15 to 30 cu. ft. capacity, hauled by horse or mule, being generally used to convey the slag from the furnace to the dump.

In most of the older plants the tap-holes are only 4 or 5 ft. above the ground-level, and the height of the slag-pot and car is, therefore, limited to this small dimension. In plants of modern construction, however, as at Butte and Anaconda, Mont., the slag-spout is of sufficient elevation to allow the use of larger pots or ladles.

At the copper blast-furnaces of the Tacoma Smelting Company, at Tacoma, Wash., a fore-hearth, 14 ft. in diameter, was used to collect the matte. The slag overflowing therefrom was formerly granulated by a stream of water, resembling the practice at the Anaconda furnaces. The ground-level, however, was only a few feet above tide-water, which required the use of a belt-elevator in order to gain sufficient height so that the granulated slag could be washed into the Sound. The water-supply was small, and there was so much trouble with the belt elevator that its use was finally abandoned. Another reason for its abandonment was the fact that the granulated slag could not be used for filling-in or building-purposes, and the company needed to reclaim a large area of ground, in order to provide for an extension of its plant. After experimenting for some time, we designed a special slag-car to run over the 18-in. gauge industrial railroad with which the works was equipped, of the following description:

As is shown in Figs. 1 and 2, the pot itself, shaped like a rectangular scoop, has the discharge-end projecting well beyond the center of the car, so that when the pot is turned and tipped the slag will be thrown well clear of the tracks. The cast-iron pot, weighing about 1,100 lb., is mounted on a cast-iron turn-table, supported by an I-beam truck, equipped with 2-in. square axles bolted to the frame, with the ends machined down for the journals. The wheels of chilled cast-iron, 11 in. in diameter, have roller bearings which are held in place by heavy

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cast-iron washers; and a wide tread. Each car complete weighs 1,510 lb. and holds 1,600 lb. of slag. The cost of each car made in the shops of the company was $75. So far they have been in use for 18 months; and, apart from the removal of a few bowls which were cracked through carelessness, no repairs have been needed. The pot is dumped quite easily, as is shown in Fig. 2.

A train of slag-cars, generally 12 in number, is drawn by an electric motor to the dump. The motor man without detaching the cars, trips the latch, revolves each pot on its turn-table to the desired position by means of a long iron-bar fitting into a socket at the back of the pot, and, by raising the bar, the pot is tipped and its contents discharged. The shell, or solidified crust, next to the sides and bottom, is released and discharged by tipping the pot with a jerk; but, if the pot be gently tipped, the shell will remain in it. Fig. 3 shows a train of slag-cars and the electric motor.

A special advantage of this shape of pot is that the cars may be run in a continuous line without any intervening space between adjacent pots. This arrangement is of great value in receiving slag from a furnace which discharges continuously. There is no spilling or slopping on the truck or track. One man can haul out to a distance of 1,000 or 1,500 ft., tip and return to the furnace, 30 of these cars per hour, which is equivalent to 576 tons of slag per day. The slag, either liquid or solid, may be dumped forward, backward, or to one side or the other of the truck; in the latter case the slag is thrown at least 2 ft. beyond the ties; and, by using rails as skids, the solid slag-cake may be landed 18 or 20 ft. from the track. The cars are connected by an automatic coupler, which holds them at a given distance apart, and enables the train to be pushed or pulled without lost motion. Moreover, the automatic coupling-device removes the danger which is present when the cars are connected by hand.

Sixty of these new cars are now being used in the lead- and copper-plants of the Tacoma Smelting Co., replacing the older and more troublesome methods formerly used.

These cars, slightly altered, are used for casting base-bullion from lead-furnaces. The dross can be separated more easily and the resultant bullion is cleaner than that made in the ordinary manner of casting.

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IN MEMORY OF
HENRY CORT
BORN AT LANCASTER 1740
INTERRED AT HAMPSTEAD 1800
TO WHOM THE WORLD IS INDEBTED
FOR THE ARTS OF REFINING IRON
BY PUDDLING WITH MINERAL
COAL AND OF ROLLING METALS
IN GROOVED ROLLS.

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The Case of Henry Cort.

BY CHARLES H. MORGAN, WORCESTER, MASS.

(Lake Superior Meeting, September, 1904.)

THE case of Henry Cort comprises: (1) the nature of Cort's inventions; (2) their value to England and to mankind; (3) the remuneration received therefor by him or his family; and (4) the suitable permanent record and recognition of his services by the representatives of the art he founded—a debt long overdue, and still unpaid.

I. CORT'S INVENTIONS.

Under this head, it is unnecessary here to enter into details. The facts have been repeatedly published,[1] and a brief outline of them will be sufficient for my present purpose.

Henry Cort, born in 1740 at Lancaster, England, became a navy broker in London about 1765, and gained from that business in about ten years something more than £20,000, which he devoted to perfecting the manufacture of iron, building a mill at Fontley, near Portsmouth, and prosecuting, between 1777 and 1783, the experiments which culminated in his two patents, one for puddling iron and the other for shaping it by rolling between grooved rolls. Careful study of the contemporary evidence, including Cort's specifications, the testimony (in act as well as word) of his business rivals, the public declarations of eminent authorities, and the history of the times, leaves no possible doubt that Cort was the real inventor of

[1] See Cort's British patents, Nos. 1351 (of 1783) and 1420 (of 1784); also, a series of articles by Thomas Webster, an eminent authority on Patent Law, in the *Mechanics' Magazine*, vol. 2, London, 1859; Percy's *Metallurgy*, "Iron and Steel," London, 1864, pp. 627–639; *Memoirs of Distinguished Men of Science*, London, 1864, p. 152; Smiles's *Industrial Biography*, London, 1897, p. 114; Facts and Proofs Collected by R. Cort, London, 1855; and a partial summary, in my Presidential Address, "Some Landmarks in the History of the Rolling-Mill," delivered before the American Society of Mechanical Engineers, at the New York meeting in December, 1900, and published in vol. xxii. of the *Transactions* of that Society.

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these two inestimable improvements. Every such invention is subject to claims of priority advanced in behalf of unsuccessful predecessors. The relative rights of such claimants to the sentimental credit, so to speak, of a given step of progress, may be difficult of adjudication; and sometimes the suggester of an idea which he never effectively executed may deserve praise and thanks for his incomplete achievement; but the principles of both English and American patent law make short work with such pretensions. That law was not instituted to reward prophetic genius or intuition. It rests upon the proposition that a man who has discovered and successfully practiced an improvement in the arts, and who might possibly keep it as a trade-secret, to die with him, shall be induced, by the grant of a monopoly for a limited term, to tell his secret completely to the public, so that, after the expiration of that term, any expert in the art concerned may be not only entitled but enabled to practice it. It is a question, not of recognizing meritorious aspiration or endeavor, but of buying for public use an actually successful device. Much credit is often fairly due and freely given to unsuccessful inventors who have, nevertheless, no rightful claim to the reward of a patent-monopoly.

In the case of Cort, Dr. Percy mentions one or two prior inventors,[1] discovered by his researches, who seem to have been working on the same lines, and to some extent with the same general ideas. But they do not seem to have put their ideas into practical use. The malicious and false testimony, denying the practical merit, and thus the patentable character of his inventions, will be briefly alluded to later. It does not deserve serious consideration here. On the whole, it may fairly be said, that beyond all honest doubt and by substantially universal acclaim, Henry Cort was the first to perfect, put into successful operation, give to the world by sufficient description, and teach to other operators, his licensees, the puddling of iron and the rolling of puddled iron between grooved rolls.

[1] As to puddling, Thomas and George Cranage (British patent, No. 851, of 1766), and Peter Onions (No. 1,370, of May 1783). Since Cort began experimenting in 1777, the date of the latter is not proof of priority of invention. There is no evidence that either of them was commercially practiced. Concerning the use of grooved rolls, I think Cort's originality is not denied.

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II. The Value of Cort's Inventions.

As I have already observed, Henry Cort accumulated a fortune of more than £20,000 as a navy broker, furnishing supplies for the British navy, at a period when England stood at bay, fighting for her colonial possessions in North America, and almost for her own national existence, and when the power and prowess of her navy was her chief, if not her only, reliance. Yet no forge or furnace in England could make iron fit for navy use. The nation, though endowed with vast deposits of iron-ore and of coal, was dependent upon the purer raw materials of Sweden and Russia for wrought-iron of good quality; and the Admiralty specifications called for Swedish or Russian iron. While Cort was in business as a navy broker, the demand for these foreign irons increased the price nearly 200 per cent. What was even worse, the dependence of England upon this foreign supply might possibly, at any moment, become a source of fatal weakness in war.

No doubt it was the perception of this critical situation which stimulated Henry Cort, both as a patriot and as an intelligent inventor, to risk his entire fortune and future upon the attempt to render his country independent of the world in a particular so essential to her prosperity. And the result of his enterprise may fairly be said to have altered profoundly the history of the world. Before 1785, England paid annually to Sweden alone £1,500,000 for wrought-iron. As the *London Times*[3] has said:

"Then came the war, commercial embarrassment, depreciated paper, foreign prohibitions, and an overpowering and increasing demand for more and more iron. The inventions of Henry Cort carried us easily through this period of sharp trial, and, as his descendants allege, were the principal cause of our success."

A single instance may show how great was the revolution effected by Cort's improvements. Richard Crawshay, one of the first of Cort's licensees, who was making under the hammer barely 10 tons of bar iron per week, increased his product by the process (concerning which he wrote, "I took it from a Mr. Cort, who had a little mill at Fontley, in Hampshire") to 200

[3] July 29, 1856.

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tons per week. But more significant still is the fact that, after the introduction of Cort's inventions, the Admiralty specifications, instead of excluding English iron, called for it.

Lord Sheffield said in 1786, only two years after the date of Cort's second patent:

"If Mr. Cort's very ingenious and meritorious improvements in the art of making and working iron, the steam-engine of Boulton and Watt, and Lord Dundonald's discovery of making coke, should all succeed, it is not asserting too much to say that the result will be more advantageous to Great Britain than the possession of the thirteen colonies of America."

And in 1865, nearly eighty years later, William Fairbairn said:

"Henry Cort's inventions have conferred an amount of wealth on Great Britain equal almost to six hundred millions sterling, and have given employment to six hundred thousand men."

The prophetic declaration of Lord Sheffield, quoted above, suggests another element in the military and commercial triumph of England which demands recognition—namely, the steam-engine of James Watt. The wonderful results of this improvement upon the Newcomen engine, first applied to hoist water from mines, have been abundantly, yet not too abundantly, celebrated. At various places in England, there are statues and memorials—half a dozen, at least—in honor of Watt; and another is projected, to be erected by British and American engineers, under the lead of Mr. Carnegie. They are not too many; he deserves them all. But it should not be forgotten that his friend Henry Cort furnished the most immediately—and, perhaps, we might even dare to say, the most permanently—important use for the steam-engine.[4]

[4] Mr. Charles H. Loring, in his Presidential Address before the American Society of Mechanical Engineers (1892), paid an eloquent tribute to the steam-engine, declaring it to be the great underlying cause of the wonderful progress of the last 100 years, and adding, "It is what no other machine ever was, the creator of physical power. The immortal inventor died without dreaming that he had placed on earth an infant Hercules, whose club, with an ever-increasing might, would batter down the institutions of preceding ages."

I cannot do better than repeat here, from my address on the history of the rolling-mill, delivered eight years later, and already cited in this paper, my comment upon this passage in Mr. Loring's address:

"I endorse that tribute, with this distinction only:—Watt's engine is the Hercules; but the rolling-mill is his club! Disarm him; take his club away—and

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For what other application of that great motor can be deemed, even to-day, more important than the manufacture of the material of which itself consists, and through which, in countless forms and ways, its benefits are conferred upon men?

It would be ungrateful, as well as unjust, in setting forth the merits of Cort, to diminish in the least the credit and fame of Watt; for the appreciation and sympathy of Watt were of value untold to Cort, in his misfortunes, though these were of a sort which Watt himself had never been called to encounter. We are now considering, however, simply the value of Cort's two great inventions, not his personal experience; and on this point what I have already said may suffice, since it has been amply corroborated by the highest authorities.[5]

III. Cort's Reward.

This chapter also I shall make as brief as possible. Not long ago, I printed, for private circulation only, a "Review of the Case of Henry Cort," which I shall be glad to furnish to those who are seriously and sufficiently interested in its details. Here I shall confine myself to a bare statement of the essential facts.

1. For the purpose of obtaining additional capital, Cort

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how little, with his vast strength, can he do! The steam-engine may be 'what no other machine ever was, the creator of physical power;' but the rolling-mill has ever been the creator of the harness for using that power."

Discarding poetic expression, we must of course confess that the steam-engine does not create, but only liberates and transforms, the power already existing, latent, in the fuel it consumes; while the rolling-mill further transforms the same power into the molecular movement and consequent change of form of the metal it treats. But such comparisons serve no useful purpose. The overwhelming, providential fact is, that just at the time when Watt's steam-engine achieved recognition, Cort gave it the means of its most stupendous achievement, in driving a rolling-mill, and thereby inaugurating the most profound industrial revolution recorded in human history. Not steam alone, but steam and iron, have wrought that revolution.

[5] In addition to Percy and Webster, already cited, I may mention here the Duke of Devonshire, in his inaugural address at the first meeting of the Iron and Steel Institute in 1869 (*The Iron and Steel Institute Transactions*, vol. i., p. 5); Sir James Kitson (*Journal of the Iron and Steel Institute*, 1890, p. 392); Sir Lowthian Bell (*Ibid.*, p. 422); Sir William Roberts-Austen (*Ib.*, 1899, p. 21), and Mr. Andrew Carnegie (*Iron Age*, October 27, 1904, p. 45),—five Presidents of the Iron and Steel Institute (and Dr. Percy makes six), the body best qualified to give judgment in such a case. In fact, there is no difference of opinion on the subject among competent experts.

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made a business agreement with one Adam Jellicoe, a Deputy Paymaster in the Navy, whose son Samuel he admitted to a partnership in his business, in consideration of which the father advanced to the firm considerable sums of money, taking as collateral security for Cort's share in the firm's debt, an assignment of Cort's patents. The arrangement was fair enough on its face; but the moneys advanced under it by Adam Jellicoe had been stolen from the government funds, with the collusion of the Treasurer, Mr. Dundas, afterwards Lord Melville. That Cort had any knowledge of this circumstance, there has never been either proof, accusation or suspicion.

2. The embezzlements of Adam Jellicoe had been carried on for years with the knowledge of the Treasurer and his Paymaster, one Trotter; but in 1788, under the pretense of a claim for repayment, these two forced him to surrender to them the Cort patents, which he held as collateral. Soon after (in 1789), Adam Jellicoe died; Lord Melville and Trotter instantly "discovered" his defalcation; and the latter, by a false affidavit, obtained an "extent" or summary attachment upon Cort's whole business and property.

3. At that time, the amount of the alleged defalcation of Jellicoe, for which Cort was thus arbitrarily made responsible, was only £27,500. Cort was then executing a Navy contract for £15,000; and was receiving in royalties £15,000 per annum. Yet Trotter swore that he verily believed the said Henry Cort to be "much decayed in his credit and in very embarrassed circumstances," so that His Majesty could not recover "the aforesaid debt" without more speedy means than the ordinary process of the court. Accordingly, Cort's property was seized.

4. After this seizure, the patents were never seen; the contracts with licensees likewise disappeared; and so far as can be discovered, no royalties were ever collected under them. These royalties alone would have repaid the whole of the alleged defalcation within two years. But nothing was done to administer the property; and Cort was thus at one blow deprived without redress of all his earthly possessions—forge, mill, patents and royalties—in a word, completely ruined.

The amount of the royalties already covered by agreements would have amounted, at the date of Cort's death, to £200,000. If Lord Melville, who confessed the possession of the agree-

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ments, collected any part of this sum, he never accounted for it to the government.

5. Cort patiently sought to obtain justice, apparently without understanding the influences at work against him. He petitioned Trotter! he petitioned the Commissioners of the Navy; he petitioned Parliament; but was treated with indifference or insult. In 1794, a petition was addressed to the Prime Minister, William Pitt, praying that Cort might be appointed to some position in one of His Majesty's dockyards, or otherwise mercifully treated. This appeal brought a pension, netting him about £150 a year. After his death, his widow received net £100 a year, and later, on her decease, the two unmarried daughters received per annum £25 6s. each, "subject to reductions," or probably about £20. And this was doled out to a man from whom the government had taken more than £200,000, to repay a defalcation of £27,500, of which he was never even accused of having any knowledge!

5. But the greatest outrage was yet to come. In 1800, Henry Cort died; and as soon as he was out of the way, and his infant children were incapable of asserting their rights, Lord Melville presented a memorial to the Lords of the Treasury, setting forth the great merit and value of Cort's inventions, and asking on that account that *he himself* should be released from £25,000 of the Jellicoe defalcation, for which he held himself responsible! This petition was promptly granted; and presumably the Cort patents were turned over to the government and cancelled.

6. In 1803, Parliament appointed a Commission of Inquiry to examine the irregularities of the Treasurer, Lord Melville, and his Paymaster, Alexander Trotter; but these two worthies burned their accounts before they could be examined. The documents in the Cort case were destroyed with the rest. Upon his impeachment and trial (the amount of the defalcation of £25,000 having been made good in 1800, as stated above), Lord Melville was acquitted by a vote of 88 to 47 peers! In other words, a high official of the British Government had stolen money from its Treasury; the Government connived in his attempt to shield himself by robbing an innocent citizen, under the forms of law, abominably misused for that vile purpose; not only the patent rights which the Government had

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solemnly covenanted to protect, but also all other property of the innocent victim, were thus stolen from him by arbitrary process and without legal redress; and finally, as a climax of this performance, the original thief was rewarded, in consideration of the great value of the plunder, of which the Government of Great Britain then became the receiver!

7. It seems justly clear, however, that the Government treasury received no pecuniary benefit from this crime. It had been robbed by Lord Melville; and it credited him with the value of the proceeds of another robbery, which he transferred to it; but, so far as can be discovered, it did not realize any part of this value by collecting from Cort's licensees the royalties they were pledged to pay. Consequently, the beneficiaries of the outrage, next to the titled criminal (and to a far greater pecuniary extent), were those who were thereby released from the payment to Cort of the royalties (amounting to $1,000,000) which they had agreed to pay him, and, next to these, the manufacturers who, during the term of Cort's patents, used his inventions without even agreeing to pay royalty, and were never called upon for such payment. In view of the rapid extension of the practice of these inventions, thus prematurely thrown open to the public in total disregard of the inventor's guaranteed rights, it is a safe estimate that $2,000,000 more was saved to somebody. Whatever may have been the injustice of the summary proceeding by which Cort was robbed and ruined, it was clearly the duty of the Government, as the holder of the patent-rights and other property which it had seized, to administer them, and account for the resulting revenue. There is not the least trace that this was ever attempted; and in view of the revealed character of Lord Melville (which the whitewash applied by the Peers cannot hide), it is hard to believe that immunity from those payments of royalty under Cort's patents, which he could have enforced, was not bought from him by bribery.

Unfortunately, there is evidence, confirming this suspicion, of shameful treachery and falsehood on the part of some of the ironmasters who apparently profited by the remission of royalties and the practical cancellation of the Cort patents. In 1812 a petition for a parliamentary grant to Cort's family, in reward for his services to his country, was presented in the House of

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Commons, with a recommendation from the Prince Regent. It was referred to a committee, of which the son of Lord Melville was a member. Two ironmasters were heard against the petition. Their evidence could have been easily overthrown; but the committee, excluding all evidence in contradiction or explanation of their statements, suddenly closed the investigation, and reported to the House that, though a considerable share of merit was due to Henry Cort, they could not satisfy themselves that it was sufficient to entitle the petitioner to a Parliamentary reward.

The testimony of the two false witnesses aroused much public indignation; and within a few weeks after the inquiry the chairman of the committee received a mass of evidence, concerning which he calmly said that "if it had been in time, the report of the committee would have been very different." Yet the children of Henry Cort were never able to get this evidence further considered. Even the recommendation of the committee of a grant to cover the expense of their futile petition for justice was ignored; and this amount was calmly charged against their penury.

8. The last chapter in this long story of wrong is the petition of Richard, the only surviving son of Henry Cort, presented to Parliament in 1856. Direct result it had none, so far as I know; indirectly, it was probably the cause of the grant to Richard Cort by Lord Palmerston, then premier, of an annuity of £50 (all previous Cort annuities, with their "reductions," having lapsed).

In connection with this revival of the question, several interesting publications appeared.[6]

IV. Our Unpaid Debt to Cort.

Samuel Smiles, in his biography of Cort, says:

"While the great ironmasters, by freely availing themselves of his inventions, have been adding estate to estate, the only estate secured by Henry Cort was the little domain of six feet by two in which he lies interred, in Hampstead churchyard."

[6] Among them, the article in the *London Times* of July 29, 1856, already cited, and the *Statement of the Claims of the Surviving Members of the Family of the late Henry Cort for National Compensation*, issued by the "Cort Committee," of which James Booth, LL.D., F.R.S., was chairman.

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During a recent visit to England, I tried to find that "little domain." The sexton of the church knew nothing about it; but after the entry of the burial had been found by examination of the records, a renewed search discovered at last the grave. The headstone was weather-worn, and the inscription upon it hardly legible. It seemed as if inanimate things had conspired to perpetuate the obscurity and neglect wrought by the malice of man.

I have had this stone cleaned, and have obtained permission from the parish authorities to place in the Hampstead church a bronze tablet in honor of Henry Cort, which is represented in the frontispiece, from a photograph of the plaster-cast. The head is of life-size.

When our English brethren are ready to erect the larger monument which Henry Cort deserves, no doubt we shall all be glad to co-operate with them. Meanwhile, one American, at least, has had the opportunity and privilege of expressing in an enduring form, for the encouragement of other meritorious, though defeated heroes, the assurance that, sooner or later, history will do them justice, and succeeding generations will not let them be forgotten.

SECRETARY'S NOTE.—The tablet in honor of Henry Cort, to which Mr. Morgan here refers, was exhibited at the Engineers' Club, N. Y. City, before being shipped to the Hampstead church, for the wall of which it was destined.—R. W. R.

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SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Roasting and Magnetic Separation of a Blende-Marcasite Concentrate.

BY H. O. HOFMAN AND H. L. NORTON, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Lake Superior Meeting, September, 1904.)

I. INTRODUCTION.

ZINC smelters in the central western States have established a very high standard of purity for blende-concentrates, viz., zinc 60, iron less than 3, and lead less than 1 per cent. The very low percentage of iron is made possible by the blende containing less than 0.5 per cent. of isomorphous iron sulphide and by the sparing occurrence of pyritic material in the large blende-deposits of southwestern Missouri and southeastern Kansas. The blende in these districts occurs in sub-Carboniferous limestone, and is associated with chert, residual clay, calcite, pyrite or marcasite and bitumen. The different minerals are coarse and therefore readily separated by mechanical washing from the blende, with the exception of pyrite and marcasite, as blende has a specific gravity of from 3.9 to 4.1; marcasite, from 4.85 to 4.90; and pyrite, from 4.95 to 5.10. The common procedure of dressing the ores[1] is to crush in a breaker, size in a trommel with 0.5-in. holes, pass the over-size through roughing-rolls and then jig the ore, now reduced to 0.5 in. maximum size, in roughing multiple-sieve jigs. The products of these are waste-tailings, side-discharge coarse concentrates, side-discharge coarse middlings and bottom-discharge fine middlings. The coarse concentrates are sent to the market. The coarse middlings are crushed in middlings-rolls and brought back into the system by feeding into the main trommel. The fine middlings are cleaned upon finishing-jigs. Any pyrite or marcasite present follows the blende and increases the percent-

[1] *Ore-Dressing*, by R. H. Richards, New York, p. 900 (1903).

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age of iron above permissible limits. Several processes[2] based upon roasting for porosity (Schullsburg, Wis.; Åmmeberg, Sweden), heating for decrepitation (Heusschen Process), or disintegration followed by screening (Lintorf, Prussia), have been used for separating blende and pyrite, but they have all been, or will be, replaced by magnetic separation. Pyrite, marcasite and blende are diamagnetic, blende may be feebly paramagnetic, the attraction depending upon the percentage of isomorphous iron sulphide it contains. In order to separate pyrite or marcasite from blende running low in iron, the former has to be made magnetic. This can be done in two ways, by converting the disulphide of iron into magnetic oxide or magnetic sulphide. In the work to be discussed both methods were tried.

II. The Ore and the Outline of the Experimental Work.

The ore used in the experiments was a coarse concentrate of about 0.5-in. size from the mines of the Alexandra Mining Co., Carterville, Mo., furnished by the superintendent, Mr. T. F. Lennan. It consisted of light-colored resinous blende (rosin jack), marcasite and chert. The chemical analysis gave Zn, 35.94, and Fe, 18.77 per cent. Calculating all the iron as marcasite, the rational analysis would be,—blende 53.57, marcasite 40.22 and gangue 6.21 per cent.

Preliminary experiments having shown that the ore as received, i.e., 0.5-in. material, was too coarse for the proposed work, an 8-mesh screen was chosen for the limiting size, as this appeared to work well and as it is the one best suited for roasting blende. In order to study the behavior of ore of smaller size, a 20-mesh screen was selected as the limit, because in crushing the 0.5-in. ore through an 8-in. sieve a very large quantity was found to be fine enough to pass through a 20-mesh screen. A series of experiments was then made on those two sizes. The operations were: 1. Crushing to required size; 2. Roasting above the ignition-point of marcasite, but below that of blende, and magnetic separation; 3. Roasting as under 2, sizing and magnetic separation of sized product; 4.

[2] *Production and Properties of Zinc*, by W. R. Ingalls, New York, p. 255 (1902). *Ore-Dressing*, by R. H. Richards, pp. 790, 880.

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Sizing crushed ore, roasting and magnetic separation of sized material; 5. Analytical determination of zinc and iron.

III. Apparatus and Methods of Analysis.

The apparatus used in the experiments were two heating-furnaces with Le Chatelier pyrometers and two electro-magnets.

An electric heating-furnace, similar to the retort furnace described by Hofman[3] in a paper read at the Lake Superior meeting of the Institute, September, 1904, served to determine the ignition-points of marcasite and blende. The roasting-operations were carried on in 5-in. roasting-dishes heated in a gas muffle-furnace, made by the American Gas Furnace Co., New York, with a muffle 12 in. long, 6.75 in. wide and 3.75 in. high. While one charge was being heated in the muffle, a dish charged with ore was warmed at the front so that it might be brought quickly to the muffle-temperature when the ore was to be roasted.

In the measurement of temperature the two leads of the Le Chatelier pyrometers, wound with asbestos thread, were held in a porcelain tube. The thermo-couple protruding at one end was about at the level of the ore-charge when the porcelain tube was lying on the floor of the muffle close to the roasting-dish. In this position the temperatures were taken. When the muffle had been brought to the desired temperature, it required very little regulating of the gas- and air-supply to hold it there.

For the magnetic separation of the roasted ore, two electro-magnets constructed for laboratory purposes, by Prof. R. H. Richards, were used, one, a single-pole; the other, a double-pole. The single-pole magnet is shown in Fig. 1. The core, *A*, a cylinder of soft Norway iron, fits into a brass tube, *B*, carrying the coils of copper wire; it is rounded at one end and screwed to a brass disk, *C*, at the other. It is enclosed by three layers of paper, *D*. There are 9,240 turns of double cotton-covered No. 21 copper wire, equal to a length of 8,100 feet. At the top the coil is held in place by the brass disk and protected by three rings of paper, *E*; at the bottom tape is wound in with the wire to prevent the coils from slipping. In winding the

[3] The Decomposition and Formation of Zinc Sulphate by Heating and Roasting, *Trans.*, xxxv., p. ——.

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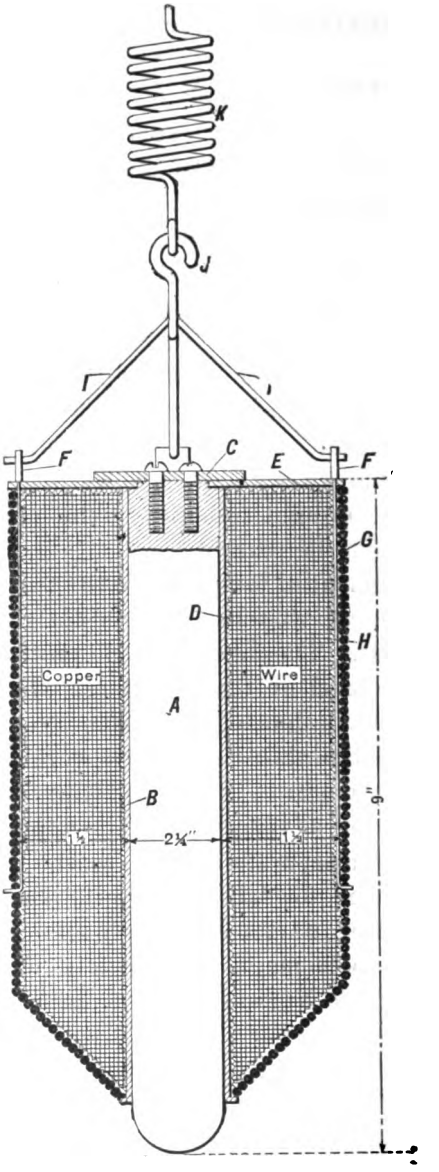
FIG. 1.



SINGLE-POLE ELECTRO-MAGNET.

wire downward, four pieces of linen tape, 4 in. long and 0.25 in. wide, are placed for a distance of 2 in. on the sides of the core. When covered with wire, the tape was bent upon itself, and the free ends now covered with wire in winding upward.

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Thus, each winding is held in place near the bottom of the core by four loops of tape and prevented from slipping. The winding is such that a current of 0.8 amperes at a pressure of 50 volts gives the maximum of magnetic lines the core can carry without undue heating. The magnet is suspended by four strips of brass, *F*, held in place by $\frac{3}{16}$-in. twine (tarred marline), *G*, wound closely around the coils previously covered with three layers of paper, *H*. The twine forms at the same time a protective casing for the magnet. Each strip has an eye near the upper end, a stout wire, *I*, is passed through it, the four wires are wound together and bent to a hook, *J*, and hung on a steel spring, *K*, suspended by an iron rod from the ceiling.

The double-pole magnet, shown in Fig. 2, has two cores of soft iron joined at the top by a yoke of cast-iron; the pole-shoes, with faces 1 by $\frac{1}{8}$ in., are screwed to the bottoms of the cores, leaving an air-gap $\frac{5}{8}$ in. long between the north and south poles. The magnet is suspended by a screw-eye from a spring in the same manner as the single-pole magnet. Each core is wound with 5,000 ft. of the same wire as the single-pole magnet, making 6,760 coils. The turns are held in place at top and bottom by brass rings and are protected by layers of paper and tarred marline. The saturation-point is the same as that of the single-pole magnet.

The attractive power of a magnet varies with the number of lines of force that pass through it. With the single-pole magnet there is the greatest scattering of lines of force, with the double-pole magnet the greatest concentration, hence for a given electric current the attraction per sq. cm. of field in a single-pole magnet is much smaller, about 60 per cent., than in the double-pole magnet, as long as the air-gap between the two poles is not too large.

The current for the magnets is taken by a shunt from the 110-volt electric-light main. In order to reduce it to the required strength, an incandescent lamp of the eight candle power is placed in the circuit. From the magnet the current passes through a Weston direct-reading ammeter back to the main. A switch is placed in the circuit to permit the current to be turned on and off.

In a test the ore was spread out on a piece of glazed paper, the current turned on, the single-pole magnet passed to and

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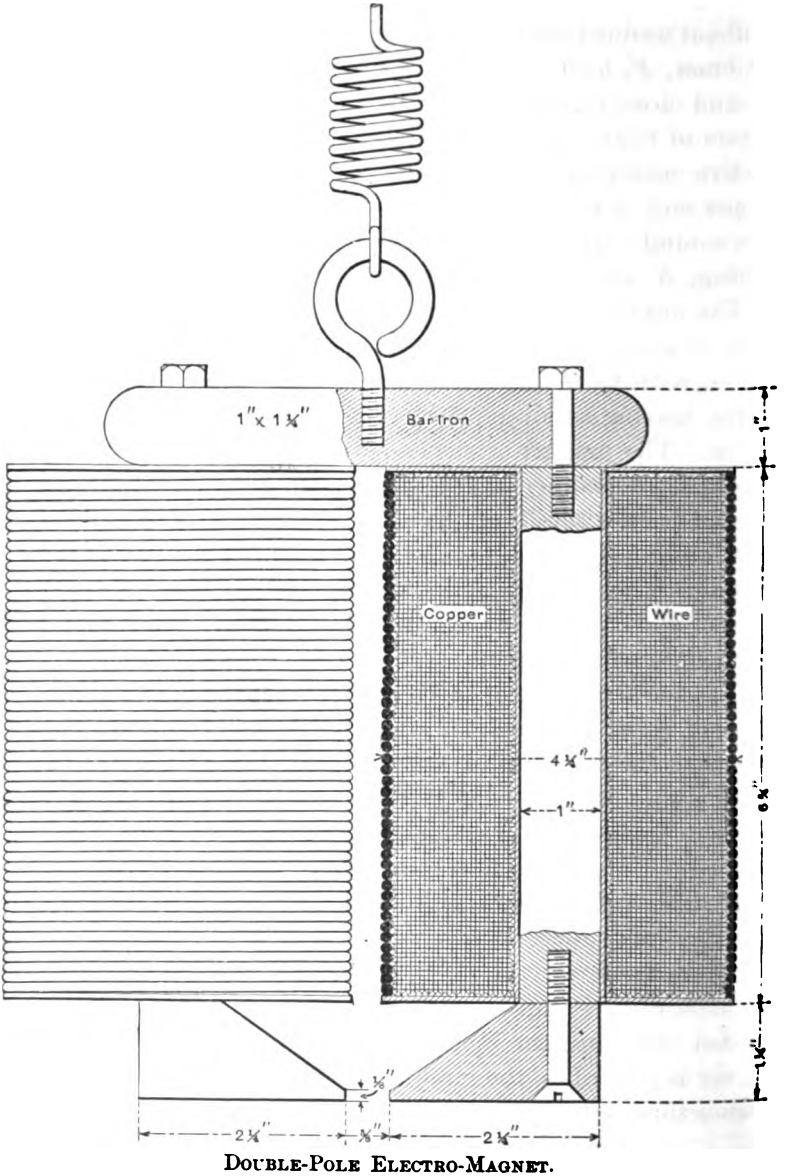
fro over the ore with raising and lowering, and then moved to one side, the current switched off and the adhering ore

FIG. 2.



DOUBLE-POLE ELECTRO-MAGNET.

dropped onto a paper. Another layer of ore was then spread over the first glazed paper, and the operations repeated until a

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batch had been treated. The first magnetic concentrates were then retreated until they had been freed from the non-magnetic particles that had been carried along mechanically. As a rule the tailings from the single-pole magnet were treated with the double-pole magnet, the manipulation being the same as in the first operation.

The zinc in the ore and the tailings was determined by titrating with potassium ferrocyanide as worked out by von Schulz and Low in 1900.[4] With the magnetic concentrates, which run high in iron and are not decomposed by treatment with nitric acid and potassium chlorate, solution with nitro-hydrochloric acid had to be substituted. The iron was converted into sulphate and titrated with potassium permanganate.

IV. Experiment with 8-Mesh Material.

Crushing.—The ore as received, of 0.5-in. size, was crushed in a Hendrie and Bolthoff sample-grinder to pass through an 8-mesh sieve. The screen-analysis of the product is given in Table I.

Table I.—*Screen-Analysis of* 8-*Mesh Ore.*

Size of Screen.	Grams.	Per Cent.
Through 8- and on 12-mesh	54	33.1
Through 12- and on 16-mesh.....	18	11.0
Through 16- and on 20-mesh.....	16	9.8
Through 20-mesh.....	75	46.0
Totals.....	163	99.9

The data in Table I. show that 46 per cent. of the ground ore was fine enough to pass through a 20-mesh sieve. It is this figure which decided the choice of 20-mesh material for the tests on fine ore.

2. *Roasting and Magnetic Separation of Unsized Material.*—Before any roasting could be undertaken, it was necessary to determine the limits of temperature which had to lie above the ignition-point of marcasite and below that of blende. The two points were found to be 378° C. for marcasite and 600° C. for blende. The first tests were planned to keep the time constant

[4] *Production and Properties of Zinc*, by W. R. Igalls, p. 99.

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and to vary the temperature. The leading facts and results are brought together in Table II.

TABLE II.—*Experiments with Unsized 8-Mesh Material.*

Roasting.									Magnetic Separation.								
										Heads.					Tailings.		
										Single-Pole.			Double-Pole.				
No. of Sample.	Raw Ore.	Depth of Charge at Center.	Roasted Ore.	Loss in Roast.	Loss in Roast.	Temperature of Muffle.	Duration of Roast.	Weight Taken.		On Weight Taken.	On Raw Ore.		On Weight Taken.	On Raw Ore.		On Weight Taken.	On Raw Ore.
	Grams.	Inches.	Grams.	Grams.	Per Cent.	Deg. C.	Min.	Grams.	Grams.	Per Cent.	Per Cent.	Grams.	Per Cent.	Per Cent.	Grams.	Per Cent.	Per Cent.
1	200		175	25	14.3	400	105	175	.....	.....	.....	4	2.2	2.0	171	97.8	85.5
2	200		168	32	19.0	410	110	168	.....	.....	.....	5	2.9	2.5	163	97.1	81.5
3	200		170	30	17.7	460	120	170	.....	.....	.....	19	11.2	9.5	151	88.8	75.5
4	200		165	35	21.3	490	125	165	7	4.2	3.5	16	9.7	8.0	142	86.1	71.0
5	200		159	41	25.9	560	120	159	12	7.5	6.0	20	6.3	10.0	127	86.2	63.5
6	200		158	42	26.6	620	120	158	17	10.8	8.5	19	12.0	9.5	122	77.2	61.0

In the six tests made to convert by long-time roasts marcasite into magnetic iron oxide, the time given to roasts Nos. 3, 5 and 6 was 120 min.; roast No. 4 was accidently continued 5 min. longer than the other three; roasts Nos. 1 and 2, serving as feelers, lasted only 105 and 110 min. respectively. The temperatures show a range of from 400 to 620° C. The rise above 600° C. was due to the heat generated in the burning of marcasite; it caused some blende to become superficially oxidized. The concentration-tests show that at a temperature ranging from 400 to 460° C., the ore is not sufficiently altered to be attracted by the single-pole magnet. With a rise of temperature the extraction begins with 3.5 per cent. and increases to the moderate figure of 8.5 per cent. when the limit of temperature set has been overstepped.

The double-pole magnet, within the range of from 400 to 460° C., draws out from 2.0 to 9.5 per cent. of magnetic material which was not acted upon by the single-pole. The higher temperatures, 460° and 620° C., give an additional yield of from 8 to 10 per cent. to that obtained by the single-pole magnet, but at best the total extraction of 18 per cent. is very unsatisfactory. Most of the ore removed by the magnet was fine;

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coarse particles could be moved about on the paper, but not made to adhere to the magnet. The results ought, therefore, to be improved by fine-crushing. The screen-analyses of the roasted products given in Table III. show that the 46 per cent. of material finer than 20-mesh size in the raw ore has been increased to about 60 per cent. in the roasted ore. This must be attributed to decrepitation. The 6 or 7 per cent. of ore which remained on the 8-mesh sieve find their explanation in the swelling of particles during the roast.

TABLE III.—*Screen-Analysis of Roasted 8-Mesh Material.*

No. of Sample.	Weight Taken.	On 8-Mesh.		On 12-Mesh.		On 16-Mesh.		On 20-Mesh.		Through 20-Mesh.	
	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
1	168	7	4.1	23	13.7	16	9.5	15	8.9	107	63.7
2	160	6	3.7	27	16 9	14	8.7	15	9.3	98	61.2
3	158	7	4.4	27	17.1	14	8.8	14	8.8	96	60.8
4	154	7	4.5	23	14.9	13	8.4	14	9.0	97	63.0
5	152	6	3.9	33	21.7	15	9.8	16	10.5	82	53.9
6	161	7	4.3	29	18.0	12	7.4	14	8 7	98	60 8

V. EXPERIMENTS WITH 20-MESH MATERIAL; UNSIZED.

1. *Crushing.*—The 8-mesh material was ground to pass a 20-mesh screen in the same manner as the original ore. The sizes obtained in the operation are given in Table IV. It shows

TABLE IV.—*Screen-Analysis of 20-Mesh Ore.*

Size of Screen.	Grams.	Per Cent.
Through 20- and on 40-mesh.....	89	45.5
Through 40- and on 60-mesh.....	34	17.0
Through 60- and on 80-mesh.....	10	5.0
Through 80-mesh.....	67	33.5
Total.....	200	100.0

that of the ore 45 per cent. was coarser than a 40-mesh, while 33.5 per cent. passed through an 80-mesh screen.

2. *Roasting and Magnetic Separation of Unsized Ore.*—As the results in Table II. had pointed to the conclusion that a higher temperature was more favorable to magnetization than a lower

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one, the former was chosen. The roasts were carried on at the constant temperature of 600° C., and the time gradually reduces from 120 to 1.25 minutes. The weight of a charge was 100 g., which gave a layer 0.5-in. deep in the center of the roasting-dish. In the roasts lasting longer than 5 min., the temperature-readings were taken every 5 min., and the ore was stirred every 2 min.; in the roasts lasting 5 min. or less, the temperature was read at the beginning and the end of the roast, and the ore stirred continuously. The appearance of the roasted ore varied with the time it had been exposed to heating; that from the long-time roasts was brownish, due to the more perfect oxidation; that from the short-time roasts black, on account of the partial conversion of iron disulphide into magnetic monosulphide.

After the completion of a roast, the ore was weighed, a sample taken for chemical analysis and the rest subjected to the single-pole and then to the double-pole magnet. A small portion of the zinc contained in the magnetic heads from the single-pole and especially from the double-pole magnet was not carried along mechanically, but was itself attracted by the magnet, showing that the very small quantity of iron sulphide contained in the light-brown resinous blende was sufficient to cause this to show some magnetic property. The results are assembled in Table V., which is best made clear by following through one test. In test No. 9, *i.e.*, 100 g. of raw ore, spread over the roasting-dish, gave a layer 0.5 in. thick in the center of the dish. The ore was roasted for 1.25 min. at a temperature of 600° C. The roasted ore weighed 95 g., thus losing 5 per cent. in weight by the operation. From the 95 g. roasted ore, 15 g. were reserved for chemical analysis, leaving 80 g. for magnetic treatment.

Table V., however, gives only the data calculated to the basis of the total weight of the roasted ore, i.e., 95 grams. The single-pole magnet took out 23.7 g., the double-pole magnet 15.5 g., giving a total of 39.2 g. of magnetic material, and leaving 55.98 g. of tailings. The percentage of zinc of the single-pole product was found by calculation: The total quantity zinc present in the raw or roasted ore was 35.94 grams. The tailings gave by analysis 55.98 per cent. of zinc; they contained therefore $0.5598 \times 55.9 = 31.29$ g. of zinc; the double-pole

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TABLE V.—*Experiments with 20-Mesh Material (35.94 Per Cent. Zn), Unsized.*

No. of Sample.	Roasting.					Magnetic Separation.													Tailings.				
						Heads.																
						Single-Pole.			Double-Pole.				Total.									
	Raw Ore.	Depth of Charge.	Roasted Ore.	Loss of Weight in Roast.	Temperature of Muffle.	Duration of Roast.	Calculated on Weight of Roasted Ore.	Zinc, Calculated.	Weight, Zinc.	Of the Zinc in the Raw Ore.	Calculated on Weight of Roasted Ore.	Zinc by Analysis.	Weight, Zinc.	Of the Zinc in the Raw Ore.	Calculated on Weight of Roasted Ore.	Weight of Zinc.	Of the Zinc in the Raw Ore.	Calculated on Weight of Roasted Ore.	Zinc by Analysis.	Weight of Zinc.	Of the Zinc in the Raw Ore.	Iron by Analysis.
	Grams.	Inches.	Grams.	Per Cent.	Deg. C.	Min.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Per Cent.
1	100	0.5	78	22	600	120	0	0	0	0	32.9	11.04	3.63	10.10	32.9	3.63	10.10	45.1	71.17	32.10	89.23	3.78
2	100	0.5	81	19	600	90	0	0	0	0	31.3	2.36	0.74	2.05	31.3	0.74	2.05	50.1	66.94	33.54	93.30	2.78
3	100	0.5	83	17	600	60	4.3	0	0	0	26.9	7.20	1.94	5.40	31.2	1.94	5.40	51.8	65.69	34.02	94.64	2.61
4	100	0.5	86	14	600	30	15.6	0	0	0	20.1	19.58	3.94	10.96	35.7	3.94	10.96	50.3	64.52	32.45	90.28	1.57
5	100	0.5	87	13	600	15	20.9	0	0	0	15.4	29.16	4.49	12.49	36.3	4.49	12.49	50.7	64.36	32.63	90.78	1.32
6	100	0.5	89	11	600	10	29.7	1.91	0.67	1.86	10.4	37.41	4.26	11.86	40.1	4.93	13.72	47.9	64.76	31.01	86.28	0.90
7	100	0.5	89	11	600	5	30.4	2.86	0.87	2.42	10.2	35.24	3.56	9.90	40.6	4.43	12.32	48.5	64.97	31.51	87.68	0.90
8	100	0.5	92	8	600	2.5	23.8	2.27	0.54	1.50	14.8	26.73	3.93	10.94	38.6	4.47	12.44	53.5	58.53	31.47	87.56	5.13
9	100	0.5	96	5	600	1.25	23.7	2.57	0.61	1.69	15.5	26.29	4.04	11.25	39.2	4.65	12.94	55.9	55.98	31.29	87.06	7.05

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magnetic product gave 26.29 per cent. of zinc and contained $0.2629 \times 15.5 = 4.04$ g. of zinc; the single-pole concentrates therefore contained $35.94 - (31.29 + 4.04) = 0.61$ g., which corresponds to $\frac{0.61}{23.7} \times 100 = 2.57$ per cent. of zinc. The distribution of the zinc in the raw ore among the three products was next calculated and found to be, in the single-pole product, $\frac{0.61}{35.94} \times 100 = 1.69$ per cent.; in the double-pole product, $\frac{4.04}{35.94} \times 100 = 11.25$ per cent., or a total of 12.94 per cent.; in the tailings, $\frac{31.29}{35.94} \times 100 = 87.06$ per cent. Thus 87.06 per cent. of the total quantity of zinc was recovered in the tailings which by analysis gave 7.05 per cent. of iron.

Reviewing the data in Table V. as a whole, it is seen that when the ore is roasted from 80 to 120 min. nothing is extracted by the single-pole magnet. As the time of roasting decreases, the ore becomes more and more magnetic until with a 5-min. roast a maximum of 30.4 g. is obtained. With roasts of shorter duration the quantity extracted decreases again. Using the double-pole after the single-pole magnet, 32.9 g. are taken out with the 120-min. roast, the quantity removed diminishes with the time of the roast, until with a 5-min. roast the minimum is reached with 10.2 grams. Reducing the time below 5 min. again increases the quantity of concentrate. The results of 10-min. roasts differ little from those obtained in roasting 5 minutes.

As far then as quantity of material extracted is concerned, a roast ranging from 5 to 10 min. gives the best results, as during this period the ore becomes sufficiently magnetic to allow a maximum to be removed by the single-pole magnet or by a double-pole magnet of lower magneto-motive force. The quality of the products, i.e., the percentage of zinc remaining in the concentrates, which is lost, and in the tailings, which is recovered, and the percentage of iron retained by the tailings, are plotted in Fig. 3 from the data in the table. The abscissa gives the duration of the roasts in minutes; on the ordinate are inscribed the percentages in four different scales to permit bringing the whole together in a single sheet and thus facilitate comparison. In roasting the ore 120 min., the single-pole magnet

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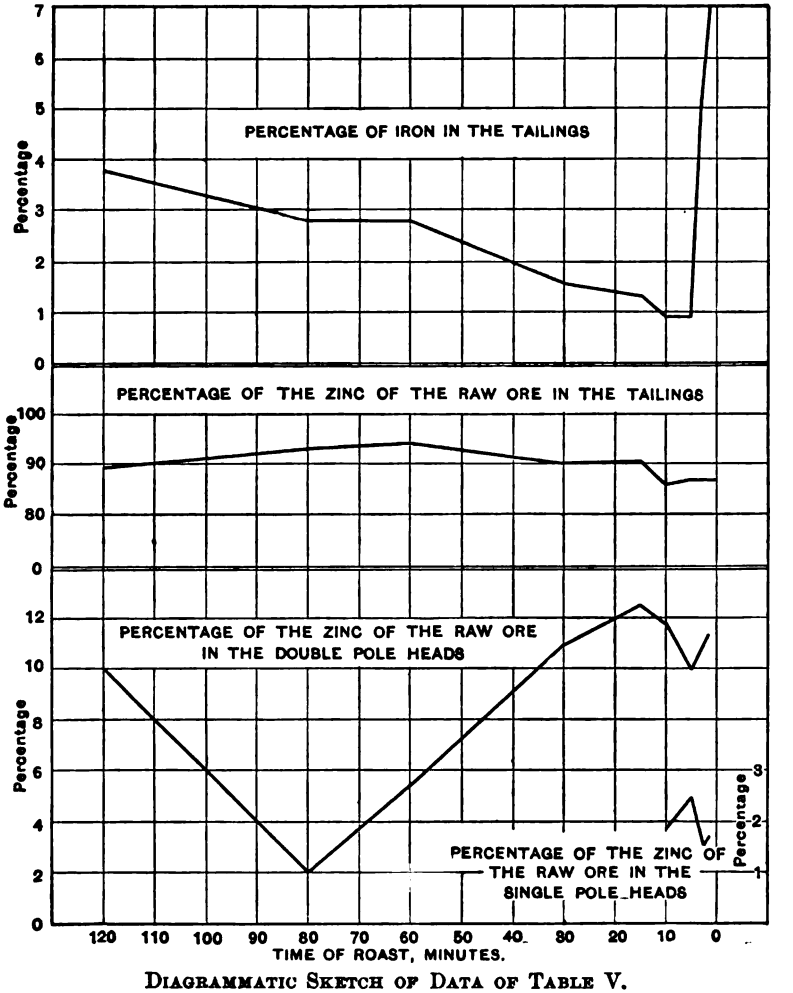
took out nothing, the double-pole concentrates carried along with them 10.10 per cent. of the zinc contained in the ore, while in the tailings there was recovered 89.23 per cent.; the tailings assayed 3.78 per cent. of iron. The quantity of zinc

FIG. 3.


PERCENTAGE OF IRON IN THE TAILINGS
Percentage
PERCENTAGE OF THE ZINC OF THE RAW ORE IN THE TAILINGS
Percentage
PERCENTAGE OF THE ZINC OF THE RAW ORE IN THE DOUBLE POLE HEADS
Percentage
PERCENTAGE OF THE ZINC OF THE RAW ORE IN THE SINGLE POLE HEADS
Percentage
120 110 100 90 80 70 60 50 40 30 20 10 0
TIME OF ROAST, MINUTES.

DIAGRAMMATIC SKETCH OF DATA OF TABLE V.

retained by the double-pole magnet decreased to 2.05 per cent. with the shortening of the time of roasting to 80 min.; it then again increased until a maximum of 12.49 per cent. was reached with 15 min.; this was followed by a second minimum of 10.2 per cent., with 5 minutes. The single-pole heads did not retain

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any zinc until the time of roasting had been reduced to 10 min., when they held 1.81 per cent. of the total quantity of zinc; at 5 min. their zinc-content reached a maximum of 2.42 per cent., which is followed by a minimum of 1.5 per cent. at 2.5 minutes.

The tailings show an approximately corresponding variation in zinc, ranging from 86.28 to 94.64 per cent., and a steadily diminishing iron-content until the 5-min. roast has been reached with 0.9 per cent. of iron, when there is a sudden increase. The most favorable result for the single-pole magnet, 1.50 per cent. of the total quantity of zinc, is obtained with a roast lasting 2.5 min.; with the double-pole magnet 80 min. give the minimum of zinc, 2.05 per cent.; the tailings furnish the highest yield in zinc, 94.64 per cent., when the roast is prolonged for 60 min., but unfortunately the percentage of iron, 2.81 per cent., is high.

Combining the four points of view, the best quality of product is obtained in roasting from 5 to 10 min.; the single-pole heads retain about 2.25 per cent. of the total quantity of zinc, the double-pole heads about 11 per cent. and the tailings collect from 86 to 87 per cent. The raw ore, with 35.94 per cent. of zinc and 18.77 per cent. of iron, has been enriched to a market blende assaying 64.8 per cent. of zinc and 0.9 per cent. of iron; and the yield of zinc is from 86 to 87 per cent.

A screen-analysis of the roasted ore, of the magnetic concentrates and of the tailings, was carried through in order to study the changes in size, etc., ore (see Table IV.) and the tailings had undergone in roasting, and to find what special sizes had been attracted by the magnets. The results are given in Table VI., in which Nos. 1 to 9 correspond to the similar designations in Table V. Comparing Tables IV. and VI., it is seen that about 3 per cent. of the ore, during the roast, increased sufficiently in size so as not to pass through a 20-mesh screen. While with the raw ore, 45.5 per cent. was caught on a 40-mesh sieve, the roasted ore yielded only 35 per cent.; the material remaining on a 60-mesh and passing through an 80-mesh sieve is about the same in both cases, 17 and 33 per cent., while that held by the 80-mesh screen is smaller with the raw ore (5 per cent.) than with the roasted ore (9 per cent.). These changes are due to the decrepitation of the blende. Of the concentrates taken out by the single-pole magnet, about 45 per

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TABLE VI.—*Screen-Analysis of Roasted, Unsized 20-Mesh Material.*

No. of Sample.	Roasted Ore.										Magnetic Heads.																Tailings.									
											Single-Pole.										Double-Pole.																	
	Weight Taken.	On 20.		On 40.		On 60.		On 80.		Through 80.	Weight Taken.	On 40.		On 60.		On 80.		Through 80.		Weight Taken.	On 40.		On 60.		On 80.		Through 80.		Weight Taken.	On 40.		On 60.		On 80.		Through 80.	
	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Grams.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
1	77	2	3	24	34	14	19	7	9	24	.....	.....	.....	.....	.....	.....	.....	.....	.....	30	9	30	5	16	3	10	13	43	41	14	34	8	19	5	12	14	34
2	75	3	4	26	34	14	18	7	9	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	29	11	37	4	13	2	7	12	41	46	19	41	7	15	4	8	16	34
3	77	2	3	27	35	14	18	7	9	27	4	.....	.....	.....	.....	.....	.....	.....	.....	25	10	40	4	16	2	8	9	36	48	17	35	10	21	5	10	16	33
4	77	4	5	27	35	16	20	7	9	23	30	14	5	36	1	7	2	12	6	43	18	7	38	3	16	1	6	7	39	45	18	35	7	15	5	11	15	33
5	79	2	2	26	33	16	20	7	9	28	34	19	9	46	4	21	1	5	5	26	14	5	35	1	7	1	7	7	50	46	17	37	7	15	5	11	17	37
6	78	3	3	29	36	15	19	6	8	25	32	26	12	46	4	15	1	4	8	30	10	3	30	2	20	.....	.....	5	50	42	15	35	9	21	4	9	17	40
7	79	3	3	29	35	15	19	5	8	26	33	27	11	41	5	14	1	4	9	33	9	3	33	1	11	1	11	4	44	43	15	35	10	23	5	11	13	30
8	81	2	2	28	34	14	17	8	7	29	35	21	10	47	3	14	1	5	7	33	13	4	30	1	7	1	7	7	53	44	18	40	9	20	5	11	12	28
9	80	2	2	29	36	15	18	7	9	26	32	20	8	40	4	20	1	5	6	30	13	4	30	1	7	1	7	6	46	46	19	41	11	23	4	9	12	27

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cent. is coarser than a 40-mesh and about 35 per cent. finer than an 80-mesh sieve. With the double-pole magnet, a little less than 35 per cent. is coarser than a 40-mesh and over 45 per cent. finer than an 80-mesh sieve. The tailings give similar figures for the larger-size material; about 35 per cent. will be caught by a 40-mesh screen, while a smaller amount of fines, 33 per cent., passes through an 80-mesh screen.

VI. Experiments with 20-Mesh Material; Roasting, Sizing and Magnetic Separation.

The aim of this series of tests was to find out whether a better magnetic separation could be obtained with sized than with unsized material, as in the preceding experiments. Some of the experiments on 8-mesh unsized material (Table I.) had shown that, while the fine particles were readily attracted by the magnet, the coarser ones could only be made to travel over the paper when the magnet was moved to and fro.

1. *Crushing.*—The ore used in the tests was taken from the same lot as that in the preceding work.

2. *Roasting, Sizing and Magnetic Separation.*—In order to have enough material of the different sizes for magnetic treatment, it was necessary to use 300 grams. The sample was, however, roasted in three separate portions of 100 g. each, in order to have the same conditions as before as far as quantity was concerned. The temperature of the muffle was maintained at 600° C., and the time given to a roast reduced to 5 min., as the previous work had shown that this gave the best results. The loss in weight in roasting was 8 per cent.

Sizing the roasted ore gave the subjoined data.

Table VII.—*Screen-Analysis of 20-Mesh Material, Roasted.*

Size of Screen.	Grams.	Per Cent.
Through 20- and on 40-mesh.....	99.3	36
Through 40- and on 60-mesh.....	53.6	19
Through 60- and on 80-mesh.....	25.5	8
Through 80-mesh.....	96.8	35
Totals.....	275.2	98

The results are similar to those of Table VI. Of the ore 36

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per cent. is coarser than a 40-mesh and 35 per cent. finer than an 80-mesh screen.

The sized material was subjected first to the single-pole and then to the double-pole magnet as described in the preceding pages. The results obtained are assembled in Table VIII., which is put together on the same general lines as Table V., with the exception that the percentage of zinc of each of the three products had to be determined analytically, as any calculation was excluded by the circumstance that the total zinc present in a mesh-size was an unknown quantity.

An examination of Table VIII. shows that a good separation was obtained with material too coarse to pass an 80-mesh screen. Thus, 40-mesh ore, which comprised 36 per cent. of the ore treated, gave tailings running 61.70 per cent. of zinc and 2.02 per cent. of iron with a loss of only 5.02 per cent. of the total quantity of zinc. Material of 60- and 80- mesh sizes gave tailings of a similar tenor in zinc with an even smaller loss in total zinc. The work upon ore finer than an 80-mesh screen gave unsatisfactory results. Forming by weight 35 per cent. of the ore subjected to magnetic treatment, the tailings could not be brought to run higher than 54.74 per cent. of zinc and lower than 5.42 per cent. of iron, accompanied by a loss in the concentrates of 10.18 per cent. of the zinc present.

The sample of 300 g. of raw ore contained (3×35.94), = 106.92 g. of zinc. Of this, 80.76 per cent. was recovered in the tailings and 19.22 per cent. lost in the concentrates, 5.16 per cent. in those of the single-pole, and 14.06 per cent. of the double-pole, the main loss being due to material finer than an 80-mesh sieve. The results are inferior to those obtained with unsized roasted ore. The plan of sizing roasted ore before treatment with the magnet is not advantageous.

VII. Experiments with 20-Mesh Material; Sizing, Roasting and Magnetic Separation.

The experiment still remaining to be made to complete the series is to size the 20-mesh ore, roast the sized products severally and then subject them to magnetic treatment.

1. *Crushing.*—The ore used in the work was taken from the main lot which had been passed through a 20-mesh screen.

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TABLE VIII.— *Work on 20-Mesh Material; Roasted, Sized and Separated.*

Weight from Table VII.		Mesh-size.	Magnetic Separation.																		Tailings.			
			Heads.																					
			Single-Pole.						Double-Pole.								Total.							
			Weight Taken.	On Weight Taken.		On Raw Ore.		Zinc.		Weight, Zinc.		Of the Zinc in the Raw Ore.		On Weight Taken.		On Raw Ore.		Zinc.		Weight, Zinc.		Of the Zinc in the Raw Ore.		On Weight Taken.		On Raw Ore.		Zinc.		Weight, Zinc.		Of the Zinc in the Raw Ore.		Iron.	
Per Cent.			Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
26		On 40.	99.3	30.6	30.8	10.2	3.58	12.29																
19		On 60.	58.6	12.1	22.5	4.0	1.90	8.32																
8		On 90.	25.5	5.1	20.0	1.7	0.47	3.82																
35		Through 80.	96.8	31.1	31.5	10.4	0.17	3.54																
Totals.							3.82	12.29				5.16												
							1.01	3.58				1.02	15.3	15.4	23.5	5.1	2.6	7.6	22.8	7.16	6.64	14.06	45.9	46.2	68.9	15.8	6.6	17.9	10.98	6.41	3.30	1.08	10.18	19.22	65.4	53.4	68.0	17.8	11.2	6.6	14.3	64.74	2.42	2.49	6.42	90.76
							0.47	3.90				0.44	7.8	14.5	14.1	2.6	1.2	7.6	3.6	0.86	0.80	2.02	19.9	27.0	34.1	6.6	2.9	17.9	4.81	2.83	0.86	7.16	6.64	14.06	33.7	68.0	40.9	11.2	6.6	14.3	64.74	2.42	2.49	6.42	90.76
							0.17	3.82				0.16	3.6	14.1	22.91	1.2		31.39					8.7	34.1	55.0	2.9		10.98		1.08					16.8	40.9	45.0	6.6		14.3	64.74	2.42	2.49	6.42	90.76
							3.82	12.29				5.16	22.8	23.5		7.6							53.9	55.0		17.9								42.9	45.0		14.3						

NOTE.—The total quantity of zinc in raw ore equals (3×35.94), 100.92 grams.

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2. *Sizing, Roasting and Magnetic Separation.*—From the screen-analysis given in Table IV. it is seen that, in order to obtain 100 g. of ore passing through a 60- and remaining on an 80-mesh sieve, 2,000 g. have to be taken for a test. The unit of ore subjected to roasting was 100 g., the temperature again 600° C. and the time 5 minutes. The roasted ore was first treated with the single-pole and then with the double-pole magnet. The results are given in Table IX.

In general character they are similar to those recorded in Table VIII., in which the roasting preceded the sizing, but denote an improvement. In the sized material coarser than an 80-mesh screen, about 92 per cent. of the total quantity of zinc is saved in the tailings which, however, run high in iron, from 2.65 to 3.30 per cent. The ore passing through an 80-mesh screen saves only 66.5 per cent. of the total quantity of zinc, and the tailings assay 8.12 per cent. of iron, which makes the method impracticable.

VIII. Conclusion.

The conclusion to be drawn from the experiments points in one direction only. Blende and marcasite in the concentrate under consideration can be successfully separated by crushing through a 20-mesh screen, roasting from 5 to 10 min. at a temperature not exceeding 600° C., and then subjecting to two double-pole magnets, the first having less than half the strength of the second.

Note by the Secretary.—Comments or criticisms upon all papers, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

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SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Blast-Furnace Plant of the "Elba" Società Anonima di Miniere e di Alti Forni at Portoferraio, Elba.

BY CARLO MASSA, GENOA, ITALY.

(Lake Superior Meeting, September, 1904.)

THE Società Anonima di Miniere e di Alti Fourni, familiarly known as the "Elba Company," works the rich ore-deposits of the Island of Elba under a long-term contract with the Italian Government. The ore-deposits are scattered over the eastern and southeastern coast; but as those parts of the island are not well provided with ports, the blast-furnaces were built at Portoferraio, the principal town of the island, situated on the northern shore, and possessing a magnificent natural harbor where even vessels as large as the battleships of the Italian navy can comfortably anchor. In the selection of Portoferraio the cost of transporting the iron-ore from the different mines to the blast-furnaces has been reduced to a minimum, and shelter has been provided for the Company's flotilla of tugs and barges, as well as for the steamers carrying coal and pig-iron.

The whole plant, of which a general view is given in Fig. 1, comprises:

1. *A discharging-pier* with unloading-cranes. The ore and coal are transported from this pier by means of an aërial cable-railway to the coal- and ore-storage yards. This cable-railway is supported by an iron bridge extending from the pier to the shore and by a wooden trestle extending from the shore to the yards.

2. *Blast-furnaces*, two in number, with all the accessory apparatus, such as blowing-engines, dry purifiers, Cowper hot-blast stoves, hoists, etc.

3. *Coke-ovens*, two batteries, with coal-bin, crushers and boilers.

4. *Engine-house* containing the blast-engines and the electric generating-station.

5. *Pumping-plant.*

6. *Narrow-gauge railways.*

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The areas covered by some of the sections of the works are:—Discharging-pier and trestles, 90,000; Blast-furnaces, purifiers, hot-blast stoves, 31,000; Coke-ovens, 42,000; and Engine-house, 19,250 sq. feet.

Electricity is extensively used throughout the works. The entire plant is lighted exclusively by electricity, there being in service about 300 incandescent lamps (of 10 and 16 candle-power) operating at 250 volts on the two-wire system, and 40 arc lamps operating at 42 volts with a current of 10 amperes. In addition to lighting, electricity is the only motive power used except for the blowing-engines, the hoists, cranes, centrifugal pumps, Theisen gas purifiers and furnace-bell being worked electrically.

A short description of the different parts of the works and some numerical data are given here below:

I. Discharging-Pier and Aërial Cable-Railway.

The entire plant, serving to unload the arriving ships and to convey the iron-ore and coal to the various parts of the works, consists of three different sections:

1. *Discharging-Pier*, properly called, which is a horizontal iron structure 340 ft. long and 53 ft. wide, with wooden platform placed at a height of about 10 ft. above the sea-level. This structure is supported by piles formed of stone and concrete, varying in depth from 43 to 56 ft. under sea-level. On this pier is placed the entire machinery necessary for working the aërial cable-railway and 8 hoisting-cranes. The latter are arranged in two groups of four placed on each side of the pier, so that the unloading of ships may proceed simultaneously on both sides. One side is reserved for the coal-steamers, the other for the iron-ore barges, and each crane can unload 25 tons per hour. The coal or iron-ore unloaded from the ships is dumped into hoppers from which the buckets of the cable-railway are filled. These buckets have a capacity of 2,200 lb. on the iron-ore side of the pier and of 1,650 lb. on the coal side. They are filled and attached automatically to the traveling-cable and run at a speed of about 4 km. (2.5 miles) per hour. The necessary motive power is furnished by three 50-h.p. Westinghouse four-pole compound-wound motors running at 900 rev. per min. and at a tension of 220 volts. Each of the two hoisting-crane

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groups is worked by one motor by means of a belt pulley and countershaft; while the third motor runs the aërial cable-railway by means of simple belt transmission.

2. *Connecting Bridge*, which is a lattice-iron structure having a slope of about 0.07 per cent., that serves to support the cable-railway and also compensates the difference between the level of the pier and that of the trestles.

3. *Trestles*, which are at the end of the connecting-bridge, support the two aërial cable-railways that branch off in two different directions, one to the ore-storage yard the other to the coal-yard, and travel over wooden trestles having a height of 52 feet. The buckets (or cars) are tripped at the proper places on to the heaps below.

II. Blast-Furnaces and Accessory Apparatus.

Originally the two blast-furnaces of the plant were built on the same pattern each of the dimensions:—Total height, 74; diameter of throat, 15; diameter of bosh, 20, and diameter of crucible, 10 feet.

Each furnace had six tuyeres and was intended to produce 150 tons per day. The normal output of each furnace, however, was 200 tons of pig-iron in 24 hours.

Later, it was decided to modify the lines of furnace No. 1, so as to increase its capacity to 350 tons. The dimensions after remodelling were:—Diameter of throat, 15; of bosh, 21, and of crucible, 12 feet. The number of tuyeres was increased to 12, which were uniformly distributed around the furnace, and a Sahlin patent spiral-cooled bosh-jacket was added. The operation of this furnace has proved satisfactory and shortly furnace No. 2 will be put out of blast and remodelled in like manner.

The two blast-furnaces work with different kinds of ore. One uses the more phosphoric ores and produces high-grade foundry-iron; while the other uses very pure ores (containing from 0.007 to 0.010 per cent. of phosphorus), and produces first-class hematite-iron.

The bells closing the hoppers at the furnace-tops are worked electrically by small series-wound motors controlled by tramway-type controllers. The raw materials—coke, ore and lime-

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stone—are hoisted electrically and distributed in the hopper by hand.

A Krupp electric crusher is used to crush the hardest ores before charging them into the furnace.

The accessory apparatus comprises:—

1. *Hoists* for raising the raw materials to the top of the furnaces.

2. *Dry purifiers* where the dust carried by the gases is deposited.

3. *Theisen gas purifiers* where these gases are washed.

4. *Cowper hot-blast stoves* which heat the cold air-blast coming from the blast-engines.

The building for the hoisting machinery, placed between the two furnaces, contains two machines, each capable of lifting in 30 seconds two loaded wagons (weighing from 1,750 to 2,200 lb. each) to the top of the furnaces. Each hoist is constituted by two drums rotating in opposite directions on which the cables supporting the hoisting-cages are wound. These drums are driven by Westinghouse four-pole series-wound railway-type enclosed motors of 50-h.p. capacity controlled by commutator-type controllers. The movement of the motors is transmitted to the drums by means of double gearing.

The furnace-gases are collected at the top of the furnace by a perpendicular tube passing through the center of the bell, and thence led through a downcomer to the dry purifiers, of which there are four for each furnace. The greater part of ore and carbon-particles carried by the gases is deposited here; but the dry purification is still carried on inside the gas-main, which is provided with dust-catchers throughout its entire length. Before being admitted to the gas-engines, the gases undergo a wet purification in the Theisen gas-cleaning apparatus and afterwards in a series of water-scrubbers where the temperature of the gas is reduced to normal. In the works there are two Theisen gas-cleaning apparatus direct-connected to Westinghouse six-pole shunt-wound motors of 100-h.p. capacity running at a speed of 520 rev. per minute.

The cold air-blast furnished by the different blowing-engines is heated in the Cowper stoves to a temperature of about 1,300° F. and then driven to the furnace tuyeres. There are four Cowper stoves for each furnace.

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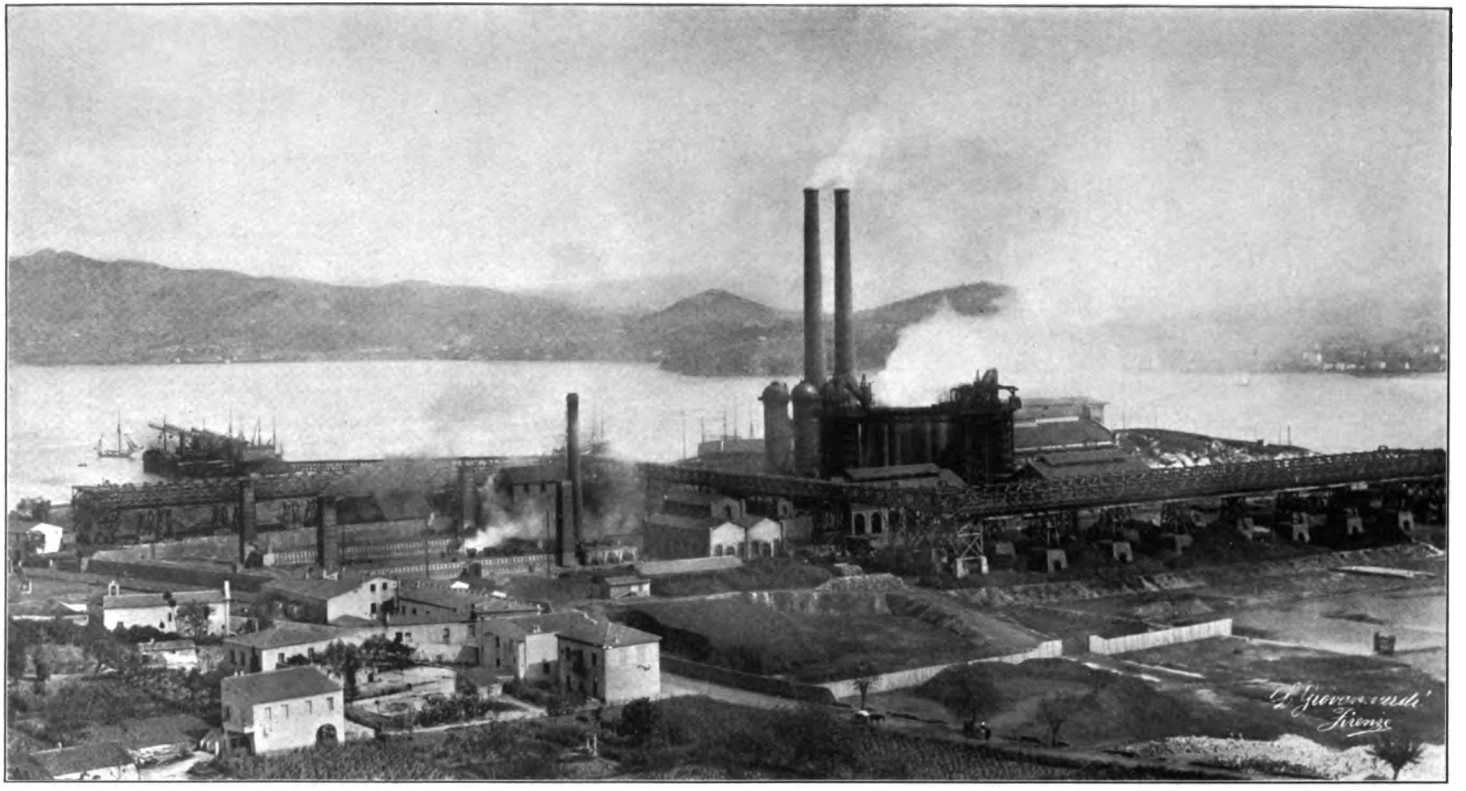

FIG. 1.—GENERAL VIEW OF THE BLAST-FURNACE PLANT AT PORTOFERRAIO, ITALY.

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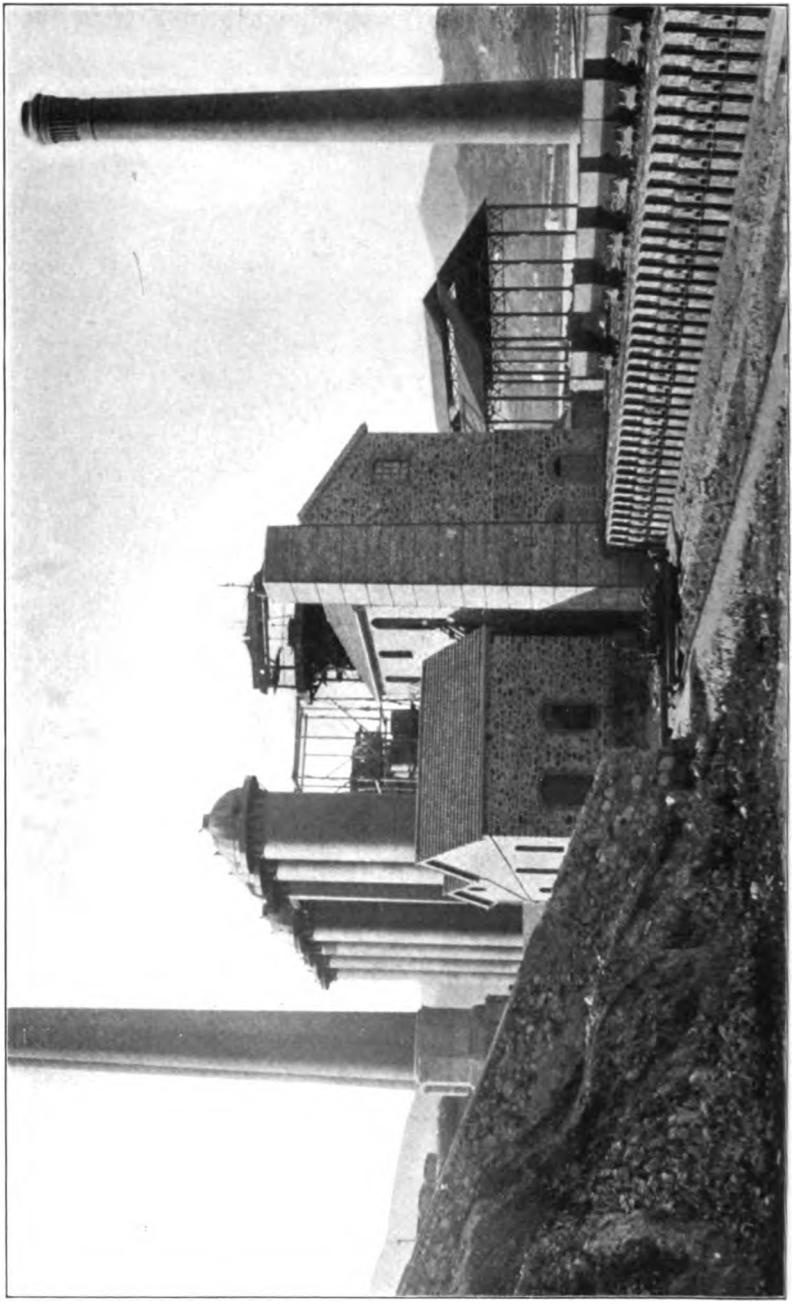

FIG. 2.—VIEW OF COKE-OVENS AND FURNACES AT PORTOFERRAIO, ITALY.

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III. Coke-Ovens.

There are two batteries, covering together an area of about 42,000 sq. ft., of coke-ovens, each containing 52 retort-ovens of the following dimensions:—length, 33 ft.; average width, 1 ft. 7 in., and height, 4 ft. 11 inches.

Each oven can yield 7,000 lb. of coke in a 24-hr. run, and the daily capacity of both batteries is equivalent to about 325 tons of coke.

To the two batteries are annexed:

1. A coal-storage bin covering a surface of 38,800 sq. feet.

2. A mill for crushing and pulverizing the coal, driven by a 75-h.p. steam-engine.

3. Three Cornwall boilers of 1,075 sq. ft. heating-surface and two fires-tubes each, using as fuel the gases produced by a set of 20 coke-ovens. The gases of the remaining ovens, up to the present, have not been utilized.

A view of the coke-ovens and boilers is given in Fig. 2.

Work will be commenced shortly on an additional battery of 80 by-product coke-ovens having an aggregate capacity of 400 tons per day.

IV. Engine-House.

The engine-house, which is a stone building 82 ft. wide and 236 ft. long, may be divided into two distinct parts: the first containing all the blowing-engines; the second containing the electric generating-station; both parts being united under the same roof.

The blowing-machinery comprises two gas-engines and one steam-engine.

Each gas-engine is of 500-h.p. normal capacity and is of the following dimensions:—Diameter of cylinder, 67 in.; stroke, 55 in., and speed, 80 rev. per minute.

The maximum output of these engines is 17,500 cu. ft. of air per min. and the pressure of blast 8 lb. per sq. inch. These gas blowing-engines are of the Cockerill-Delamare type.

The steam blowing-engine, built by the Meuse Works of Liege, is a 1,200-h.p. horizontal compound-condensing engine of the following charactertstics:—Diameter of blowing-cylinder, 84 in.; of high-pressure steam-cylinder, 39 in.; of low-pressure steam-cylinder, 63 in.; the stroke is 59 in.; the initial pressure is 132 lb.; the maximum speed is 42 rev. per minute.

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The air-output is 750 cu. ft. per revolution. At that rate the blast-pressure can be raised as high as 10 lb. To this engine is annexed a sea-water cooled surface-condenser with a cooling-area of 2,400 sq. ft. having the necessary pumps.

A fourth gas blowing-engine now being erected is a 1,500-h.p. Cockerill-Delamare double-effect gas blowing-engine, of the following proportions:—Diameter of gas-cylinder, 51 in.; of air-cylinder, 87 in.; the stroke is 55 in.: the speed is 80 rev. per min., and the air-output at 12-lb. pressure is 28,000 cu. feet.

The electric generating-station consists of three belt-driven continuous-current 100-kw. dynamos connected to gas-engines, and one belt-driven continuous-current kw. dynamo connected to one of the celebrated Tosi engines.

Each of the three gas-engines, of the Cockerill system, has a capacity of 200 h.p. and is of the following proportions:—Diameter of cylinder, 33 in.; stroke, 39 in., and speed, 105 rev. per minute.

The steam-engine, built by the Tosi firm, of Legnano, is a horizontal compound-condensing engine with tandem cylinders (Woolf engine); its capacity is 280 h.p. normal and 370 h.p. maximum. Its principal data are: Diameter of high-pressure cylinder, 18 in.; of low-pressure cylinder, 30 in.; the stroke is 41 in., and the speed, 80 rev. per minute. Each engine is provided with a sea-water cooled surface-condenser and the necessary pumps.

The four dynamos, of Westinghouse construction, are all six-pole compound generators. The speed of the 100-kw. units is 550 rev. per min., that of the 200-kw. unit, 450 rev. per minute. They generate current at a tension of 250 volts, which is the general voltage of the entire continuous-current distribution. There are two mains only.

The ignition-circuit of the gas-motors is fed from two small motor-generators reducing the voltage from 250 to 4 volts; the the 250-volt bi-polar-motors of 0.5-h.p. capacity are direct connected to the 0.33-h.p. dynamos generating-current at 4 volts; the common speed is 1,800 rev. per min., and a small storage-battery is used as reserve to these sets. The continuous current thus generated is rendered discontinuous in the gas-engines by Ruhmkorff coils.

The engine-house still contains an electric transforming

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sub-station where the low-tension continuous-current is converted into high-voltage alternating-current and transmitted over a line of two miles to a pumping-plant which supplies water to the boilers. This sub-station consists at present of a continuous-current four-pole 30-h.p. shunt-motor direct-connected to a six-pole three-phase generator of 15-kw. capacity generating-current at 1,100 volts; the speed of the set is 975 rev. per minute.

An additional transforming-group, to be installed shortly, consists of a 50-kw. separately-excited inverted rotary-converter connected to three step-up transformers. The transmission-line voltage is 1,200. The engine-house is spanned by a 20-ton traveling-crane.

The steam for the steam-engines is supplied by a set of three multitubular Tosi boilers heated by purified blast-furnace gas. Each boiler has a heating-surface of 2,850 sq. feet.

V. Pumping-Plant.

The pumping-plant is of two different sections; the cooling-water section and the feed-water section.

The cooling-water plant is situated on the sea-shore at the works, and contains four groups, each consisting of a Westinghouse 60-h.p. four-pole shunt-motor direct-connected to two Dumont centrifugal tandem-connected pumps. Motors and pumps are placed on separate bed-plates. The output of each group at a speed of 890 rev. per min. is 8,750 cu. ft. per hr.; the water is raised to a height of 99 ft. above the sea-level and stored in two metallic tanks of 8,750 cu. ft. capacity.

The sea-water is used for bosh- and tuyere-cooling and for the cylinders of the gas-engines. For cooling the ignition-chambers of the latter, the same water is supplied at a slightly higher pressure by two small steam-pumps drawing water from the discharge-pipe of the tanks.

The section of the pumping-plant which supplies feed-water to the boilers is situated at a distance of about 2 miles from the works. The water is obtained from three wells about 50 ft. deep. A reciprocating-pump driven by a high-tension induction-motor raises the water and drives it to a tank of 19,250 cu. ft. capacity placed 230 ft. above the pumps. This tank is built

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partly of stone and partly hewn in the rock and from it the water runs directly to the works.

A large pumping-group, corresponding to the increased needs of the works, now being erected, will shortly be put in operation. It consists of a Westinghouse induction-motor of 40-h.p. capacity running at a speed of 720 rev. per min. with a tension of 1,000 volts. This motor is belt-connected to a Bopp & Reuther horizontal pump having three plungers, and capable of raising 2,450 cu. ft. of water per hr., and the dimensions of the pump are:—Diameter of plungers, 5.1 in.; the stroke is 7.9 in., and the speed is 156 rev. per minute. There is also a reserve steam-pump to which steam is supplied by a Cornwall boiler of 240 sq. ft. heating-surface.

VI. Railways.

In addition to the above-described machinery, etc., there are two small railway-lines within the works. One, a 47-in. gauge track, serves to carry the slag-ladles to and from the furnaces. The slag-ladles are hauled by an electric-trolley locomotive equipped with two 13-h.p., 220-volt series motors. The second railway is a steam-line of 31.5-in. gauge on which two locomotives haul the limestone and pig-iron cars to and from the blast-furnaces.

Note by the Secretary.—Comments or criticisms upon all papers, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

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SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Effect of Silver on the Chlorination and Bromination of Gold.

BY H. O. HOFMAN AND M. G. MAGNUSON, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Lake Superior Meeting, September, 1904.)

INTRODUCTION.

WHEN dry chlorine gas is made to act in the cold upon finely-divided gold,[1] it converts the latter with evolution of heat into auro-auric chloride, Au_2Cl_4, a hard, dark-red, hygroscopic salt. Moisture splits this salt into aurous and auric chloride, $Au_2Cl_4 = AuCl + AuCl_3$; treatment with water converts it into auric chloride and gold, $3\ Au_2Cl_4 = 4\ AuCl_3 + Au_2$. Aurous chloride, when stirred with water, undergoes a similar decomposition, $3\ AuCl = AuCl_3 + Au_2$. These decompositions of auro-auric chloride and of aurous chloride furnish the explanation for the practice of moistening an ore before it is treated by the Plattner chlorination-process; practical experience having shown that gaseous chlorine gave an unsatisfactory extraction with dry ore.

According to Rose,[2] fine gold is acted upon more slowly by chlorine than gold containing some base metal, e.g., copper. He also states that small quantities of silver increase the rate of solution, but adds that the coating of silver chloride formed checks, and finally stops, further action, if the percentage of silver be increased beyond a certain undetermined amount.

Bromine[3] acts upon finely-divided gold in a manner similar to chlorine; the auro-auric bromide, Au_2Br_4, however, is not hygroscopic. Upon treatment with water it is decomposed as is the corresponding chlorine salt. It is generally believed that bromine does not act as energetically upon gold as does chlorine. This seems to be borne out by the results in the leach-

[1] Thomsen, *Journal für Praktische Chemie*, xiii., p. 337 (1876).
[2] *Metallurgy of Gold*, by T. K. Rose, London, p. 268 (1902).
[3] Thomsen, *Op. cit.*

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ing plant of the Black Hills Milling and Smelting Co., Rapid City, S. D.,[4] where barrel-chlorination and bromination were compared in the laboratory and in the mill, and chlorine was found to make poorer tailings than bromine. Rose,[5] however, in experimenting upon the rate of solution of gold by chlorine and bromine, found that bromine dissolved gold more rapidly than chlorine, and that the action of both was quicker at from 50° to 60° C. than at ordinary temperature.

Percy[6] describes two striking lecture-experiments which show how readily silver leaf is converted into silver chloride or bromide by the action of gaseous chlorine or chlorine water, or by bromine vapor or bromine water. A large bottle is filled loosely with silver leaf; upon introducing chlorine or bromine the leaf is entirely converted in a short time into white chloride or yellowish bromide. As to the effect of silver upon chlorination of gold, the only figures published are those of Dietzsch,[7] Wagemann[8] and Coignet.[9] Dietzsch and Wagemann say that the gold must be at least 0.917 fine, if a satisfactory extraction is to be obtained, and Coignet asserts that gold with from 10 to 12 per cent. of silver is more readily dissolved than pure gold, but that gold with from 40 to 50 per cent. of silver cannot be any more treated by chlorine with a view of dissolving the gold.

In regard to the effect of silver on bromination of gold, no data appear to have been published.

In chlorinating gold with gaseous chlorine in a stationary vat, the gold should be more effectively protected by the silver chloride formed than when chlorine-water or bromine-water is used in a revolving barrel, as the coating of silver chloride or bromide formed will be removed more or less by the abrasive action of the charge upon the metallic particles.

The aim of the research was to find the dissolving power of chlorine-water and of bromine-water upon gold and upon a

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[4] Godshall, *Engineering and Mining Journal*, Jan. 6, 1894.

[5] *Op. cit.*, pp. 267, 268.

[6] *Metallurgy of Silver and Gold*, by John Percy, London, Part I., pp. 56 and 110 (1880).

[7] *Berg- und Huettenmaennische Zeitung*, xl., 13 (1881).

[8] *Berg- und Huettenmaennische Zeitung*, xlviii., 249 (1889).

[9] *Bulletin de la Société de l' Industrie Minérale*, xii., 762 (1898).

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series of alloys of gold and silver, the operations being carried out in revolving vessels.

The Ore-Charge.

The ore used in the experiments was made up of quartzite and gold, resp. gold-silver alloy. The quartzite was crushed to pass a 40-mesh sieve and freed from particles of iron from the crushing-machinery by means of a magnet, and by boiling with acid. The alloys were prepared from chemically pure gold and silver. The metals were rolled into strips, the desired quantities then weighed out to 0.01 mg. and alloyed by fusing on charcoal before the blow-pipe. The resulting globules were hammered to thin disks and each converted into a fine powder by holding in pinchers and rubbing the edge on a fine file, using a very slight pressure. Any particles appearing to be coarse under a magnifying glass were put aside. The quantities used were too small to allow the use of a limiting sieve; the size of particles is therefore not definitely given. A charge was made up of 5 assay-tons of 40-mesh quartzite and 10 mg. of metal, making thus an ore assaying 2 oz. per ton. The composition of the series of alloys tested was—

	mg.	mg.	mg.	mg.	mg.
Gold	10	9	8	7	6
Silver	0	1	2	3	4

The Apparatus.

The dissolving-tests were made in pint fruit-jars of glass, 3 in. in diameter and 6 in. high, the covers being held in place by screw-clamps, and the joints made tight by rubber gaskets. The jars were rotated in the Richards revolving apparatus shown in Figs. 1 and 2.[10] This is seen to consist of a horizontal shaft with pulley, having at either end a cylindrical box holding 7 jars. A box has a wooden bottom, 12 staves connected by a stout wire, and sides of 2.5-lb. lead. A jar is held loosely in a horizontal position by three sheet-steel springs screwed to the wooden bottom. A circular felt pad, *b*, tacked down between three springs protects the bottle against jarring. The barrel makes 4.8 rev. per minute.

[10] *Ore Dressing*, by R. H. Richards, vol. ii., p. 1157.

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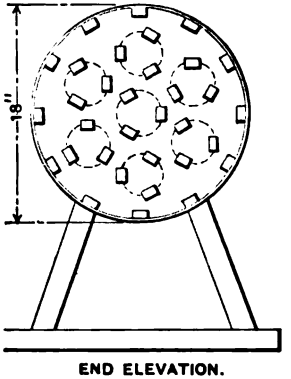
The Solvents.

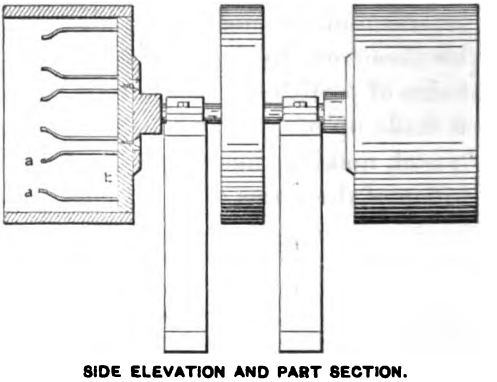
In order to obtain a satisfactory extraction of gold by chlorination or bromination, it is essential to have an excess of reagent over that taken up by the ore. In working on a large scale, it has been found[11] that the consumption of chemicals per ton of ore is the larger, the smaller the charge. Bringing this down to a laboratory-scale, the quantities necessary will be much greater than those found in practice. Considering further the manner of preparing the finely-divided gold or gold-silver, which was coarser than that found in ores subjected to leaching, the excess of solvent in the present case has to be larger than that required by a natural ore in a laboratory-test.

Fig. 1.



END ELEVATION.

Fig. 2.



SIDE ELEVATION AND PART SECTION.
a, wire springs for holding the bottle.
b, place for protective felt pad.

Richards Amalgamation Revolver.

In comparing chlorine and bromine as solvents, it will be necessary to have equivalent quantities. Guided by experiences in regular laboratory practice,[12] the quantities of bromine settled upon, using 150 c.c. of water and 5 A. T. of ore, were 3.0, 2.0, 1.5, 1.0, 0.8, 0.7 and 0.5 c.c. According to Roozeboom,[13] a saturated solution of bromine in water contains at 0° C., 4.05 per cent. of bromine; at 3°, 3.8 per cent.; and at 10°, 3.33 per cent. The solubility decreases slowly with the temperature.

[11] Godshall, *Engineering and Mining Journal*, Jan. 6, 1894, p. 6.

[12] *Notes on Assaying*, by R. W. Lodge. Wiley & Sons, New York, p. 248 (1904).

[13] *Dictionary of Chemical Solubilities*, by Comey. Macmillan & Co., London (1896), p. 62.

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Dancer[14] gives for 15° C., 3.226 per cent. of bromine, and for 30°, 3.126 per cent. With 1 c.c. of bromine weighing 2.99 g., the equivalent of chlorine will be 1.32 g., according to the ratio—Br : Cl = 79.96 : 35.45 = 2.99 : x. The chlorine in the experiments was obtained by the action of sulphuric acid upon bleaching-powder. The available chlorine in this was determined[15] by adding to a semi-solution of a weighed quantity of bleaching-powder an excess of a standardized solution of arsenious oxide, titrating back with a standard solution of iodine and calculating the available chlorine from the oxidized arsenious oxide. The bleaching-powder contained 20.78 per cent. of available chlorine, or the weight of the desired amount of chlorine in grams has to be multiplied by 4.81 in order to obtain the necessary bleaching-powder. Multiplying by 5 gives a slight excess. For each gram of bleaching-powder used, 1.5 g. of concentrated sulphuric acid (sp. gr., 1.84) was added. The solubility of chlorine in water is limited. According to Gay-Lussac :[16]

TABLE I.—*Solubility of Chlorine in Water.*

1 c.c. = 1 g. Water Dissolves at.[x]	Volume.[y]	Weight.[z]
Degrees Centigrade.	C.c.	Gram.
0	1.43	0.0045
8	3.04	0.0096
10	3.00	0.0095
17	2.37	0.0075
100	0.15	0.0002

[x] Only part of temperatures given in the original are reproduced here.
[y] Not corrected to 0° C. and 760 mm.
[z] C.c. Cl. × 0.00317344 = grams Cl.

MODE OF OPERATING.

In making up a charge, the gold- or alloy-powder was well mixed with the 40-mesh quartzite and charged into the flask. In brominating, 150 c.c. of water was added and then the required cubic centimeters of bromine dropped from a graduated pipette keeping the tip under water. In chlorinating, the

[14] Comey, *op. cit.*
[15] *Quantitative Chemical Analysis*, by H. P. Talbot. Macmillan, New York (1899), p. 111.
[16] Comey, *op. cit*, p. 105.

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bleaching-powder was mixed with the ore, the mixture filled into the flask, 150 c.c. of water added, and lastly the sulphuric acid. Although there is some danger of loss of chlorine in this

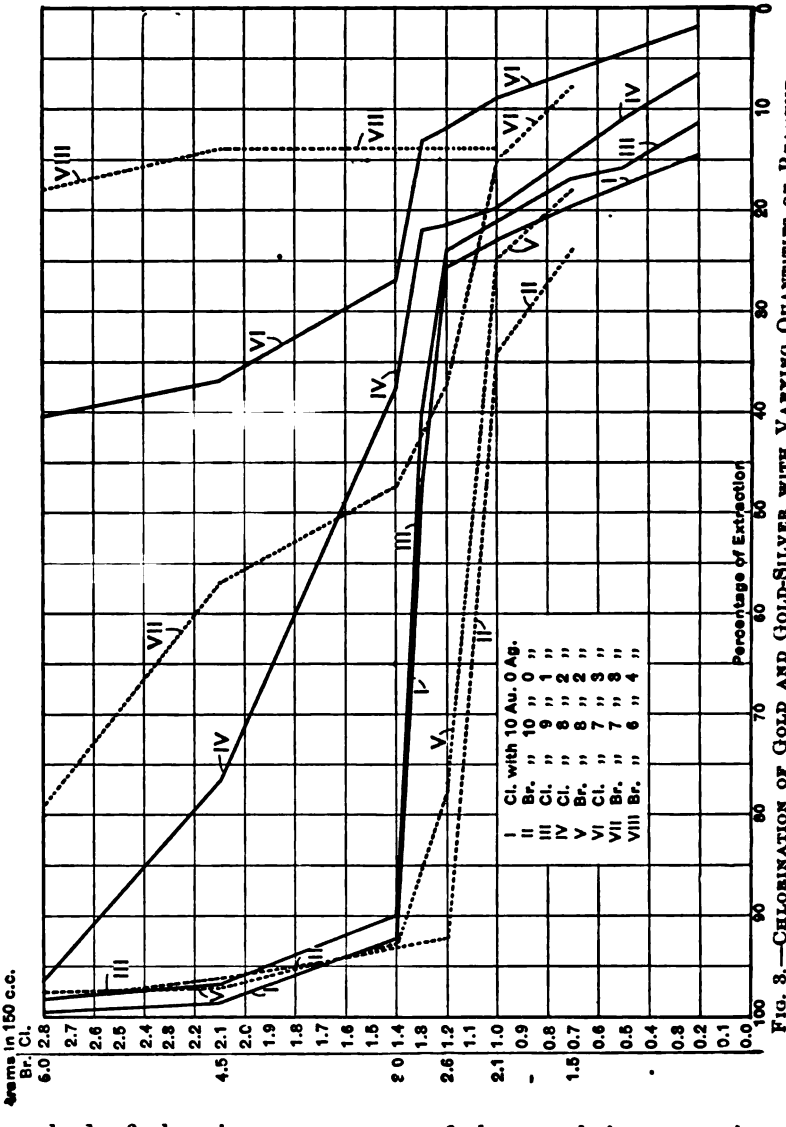
Percentage of Extraction
Cl. with 10 Au. 0 Ag.
I = Br. ,, 10 ,, 0 ,,
II = Cl. ,, 9 ,, 1 ,,
III = Br. ,, 8 ,, 2 ,,
IV = Cl. ,, 8 ,, 2 ,,
V = Br. ,, 7 ,, 3 ,,
VI = Cl. ,, 7 ,, 3 ,,
VII = Br. ,, 6 ,, 4 ,,
VIII = Br. ,, 4 ,, 6 ,,
grms in 150 c.c.
Br.
Cl.

FIG. 3.—CHLORINATION OF GOLD AND GOLD-SILVER WITH VARYING QUANTITIES OF REAGENT.

method of charging, an account of the gas being sometimes evolved before the lid can be secured down firmly, it is to be preferred to charging the bleaching-powder in bulk and then covering it in the ore, as the powder is not so liable to form

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lumps coated with calcium sulphate, which prevents further action of the sulphuric acid and thus gives less chlorine than called for by the test. The time of rotation was uniform at 5.5 hr., the barrel making 4.3 rev. per minute. When the time has elapsed, the content of a jar was transferred to a 7-in. filter and washed for from 2 to 3 hr. with from 800 to 1,000 c.c.

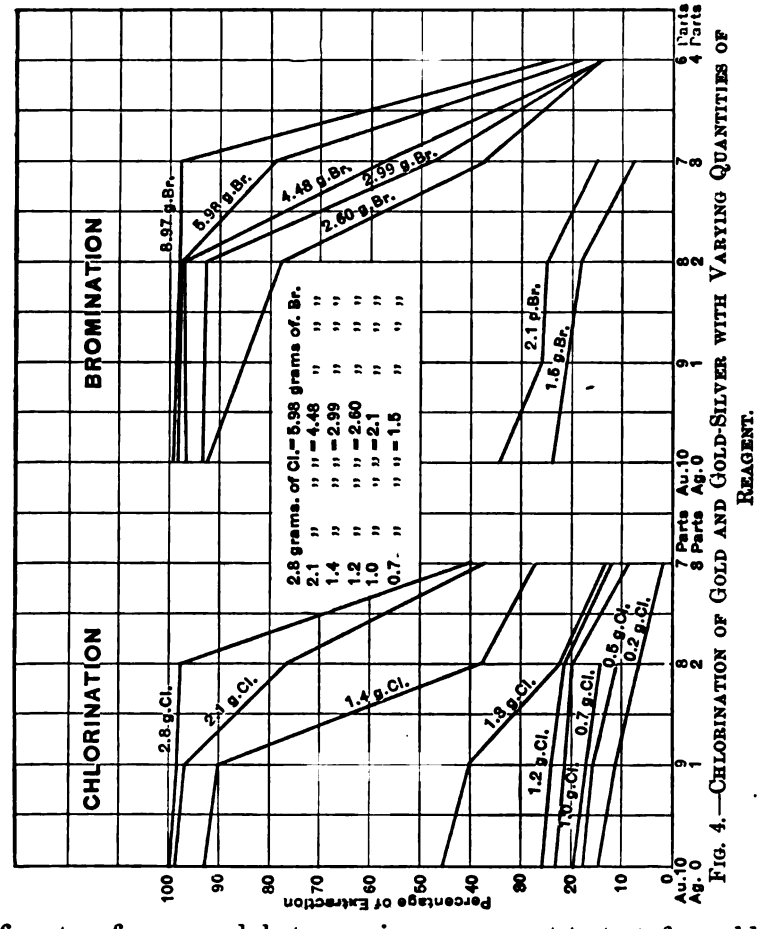

FIG. 4.—CHLORINATION OF GOLD AND GOLD-SILVER WITH VARYING QUANTITIES OF REAGENT.

of water, ferrous sulphate serving as reagent to test for gold; the residue and filter were placed on a roasting-dish at the mouth of a muffle, dried, the filter burned, the whole passed through a sieve to break up all lumps and charged with the necessary fluxes into a Battersea crucible, mark K, and fused. The charge, comprising ore, 5 A. T.; sodium bicarbonate, 150 g.; litharge, 300 g.; argols, 10 g.; and the salt cover, gave in 1.5 hr.

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a lead button weighing about 125 g., which had to be reduced by scorification to about 25 g. before the cupellation. This was done in a 3.5-in scorifier necessitating one pouring-off of slag. The alloy-buttons were parted in the usual way after adding the required quantity of silver, and fusing before the blow-pipe. The conditions of assay were kept as uniform as possible, in order that any losses by slagging, volatilization and cupel-absorption might be the same, and the results correct in relation to one another.

Results.

The results obtained are given in Table II. for chlorination, and Table III. for bromination, and graphically represented in Figs. 3 and 4 respectively. The tables require no further explanation. In Fig. 3 the constant is the quantity of active reagent, the variables are the percentage of extraction drawn as ordinate and the ratios of gold and silver drawn as abscissa. In Fig. 4, which was plotted to bring out in a more striking way the dissolving-effects of chlorine and bromine, the variables are quantity of reagent drawn as ordinate and the percentage of extraction drawn as abscissa, while the composition of the alloy is the constant. In analyzing the results of chlorination in Fig. 3, it is seen that with pure gold the solutions containing 2.8, 2.1 and 1.4 g. of chlorine give good extractions, i.e., 99.6, 98.6 and 92.5 per cent.; with 1.3 g. chlorine there is a sudden fall to 45.2 per cent., and with 1.2 g. a second decided lowering of extraction to 25.6 per cent.; from now on the yield decreases with the diminishing quantity of chlorine present in more regular way.

Observations made with the flasks during the tests are recorded in Table IV.

Gay-Lussac (see Table I.) has shown that under atmospheric pressure and a temperature of 17° C. 1 c.c. of water can dissolve only 0.0075 g. of chlorine. In order to obtain the supersaturation required for a good extraction, it is necessary to have a "strong" pressure in the revolving-jar. When the pressure falls to "medium" there is a decided diminution in the yield of gold. This agrees with the experiences made in practice where it has been found that pressure of free chlorine is absolutely necessary for a satisfactory extraction, i.e., one of over 90 per cent. As the solution of chlorine in water, according to

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TABLE II.—*Chlorination of Gold and Gold-Silver with Varying Quantities of Reagent.*

Ore, 5 A. T.; water, 150 c.c.; time, 5.5 hr.

Bleaching Powder.	Sulphuric Acid.	Chlorine (Calculated).	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.
Grams.	Grams.	Grams.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.
14	21	2.8	Gold 10 : Silver 0.	0.04	99.6	Gold 9 : Silver 1.	0.18	98.2	Gold 8 : Silver 2.	0.30	96.3	Gold 7 : Silver 3.	4.18	40.3
10.5	15	2.1		0.14	98.6		0 30	96.7		4.98	37.8		4.40	37.1
7	10.5	1.4		0.75	92.5		0.90	90.0		5.90	26.5		5.10	27.1
6.5	9.75	1.3		5.48	45.2		5.37	40.3		6.23	22.1		6.08	13.2
6	9	1.2		7.44	25.6		6.84	24.0		6.28	21.5		6.15	12.1
5	7.5	1		7.70	23.0		7.09	21.2		6.42	19.8		6.36	9.1
3.5	5.25	0.7		8.06	19.4		7.48	16.9		6.85	14.4		n. d.	.....
2.5	3.75	0.5		8.26	17.4		7.59	15.7		7.12	11.0		n. d.	.....
1	2	0.2		8.56	14.4		7.99	11.2		7.48	6.5		n. d.	.....

TABLE III.—*Bromination of Gold and Gold-Silver with Varying Quantities of Reagent.*

Ore, 5 A. T.; water, 150 c.c.; time, 5.5 hr.

Bromine.	Bromine.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.	Precious Metal.	Gold in Tailings.	Gold Extraction.
C.c.	Grams.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.	Mg.	Mg.	Per Cent.
3	8.97	Gold 10: Silver 0.	0.08	99.2	Gold 8 : Silver 2.	0.18	97.8	Gold 7 : Silver 3.	0.16	97.7	Gold 6 : Silver 4.	4.56	24.0
2	5.98		0.18	98.2		0.20	97.5		1.48	78.9		4.92	18.0
1.5	4.48		0.34	96.6		0.24	97.0		3.00	57.1		5.18	13.7
1	2.99		0.68	93.2		0.58	92.8		3.70	47.2		5.16	14.0
0.8	2.60		0.78	92.2		1.88	77.6		4.40	37.1		5.16	14.0
0.7	2.10		6.58	34.2		6.02	24.8		5.95	15.0		n. d.	.....
0.5	1.5		7.62	23.8		6.56	18.0		6.46	7.7		n. d.	.....

TABLE IV.—*Observations Made During Chlorination-Test.*

Chlorine in 150 c.c. of Water.	Extraction of Gold.	Color of Solution.	Odor of Solution.	Pressure of Solution.
Grams.	Per Cent.			
2.8	99.6	Deep yellow.	Very strong.	Strong.
2.1	98.6	Deep yellow.	Strong.	Strong.
1.4	92.5	Deep yellow.	Medium.	Strong.
1.3	45.2	Yellow.	Medium.	Medium.
1.2	25.6	Yellow.	Medium.	Medium.
1.0	23.0	Pale yellow.	Slight.	Weak.
0.7	19.4	Pale yellow.	None.	None.
0.5	17.4	Almost colorless	None.	None.
0.2	14.4	Colorless.	None.	None.

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the law of Henry, increases with the pressure, one might be led to believe that very high pressures of free chlorine would increase the extraction of the gold, but this has been found not to be the case, as, e.g., at Deloro, Canada,[17] no better results were obtained with 10- than with 60-lb. pressure. The essential point was to have at first sufficient pressure of chlorine (10–15 lb.) that at the end of the operation (after 2 hr.) there might be some left (3 lb. and less) to maintain the supersaturation of the solvent. The experimental data and results from large-scale work show that a satisfactory extraction of gold in barrel-chlorination can be obtained only by having supersaturated solution of chlorine in water; this requires a certain pressure, varying with the character of the ore, and has to be determined for every case.

In chlorinating the alloy of 9 parts gold with 1 part silver, the extraction with 2.8 g. of chlorine falls 1.4 per cent.; with 2.1 g., 2.1; with 1.4 g., 2.5; and with 1.3 g., 4.9 per cent.; the decrease in yield growing with the diminishing grams of chlorine. With smaller quantities of chlorine, from 1.2 to 0.2 g., there is no such regularity.

With the alloy of gold 8 parts and silver 2 parts, the supersaturated solution with 2.8 g. of chlorine shows only a small decrease of 0.7 per cent. in the gold dissolved. The solution with 2.1 g. of chlorine has its dissolving power diminished by 20.1 per cent., that with 1.4 g. of chlorine by 52.2 per cent.; that with 1.3 g. of chlorine falls off less, viz., 18.2 per cent.; it loses its previously isolated position and unites with the extremely low extractions of solutions with 1.2–0.2 g. chlorine.

With the alloy of gold 7 parts and silver 3 parts, the solution with 2.8 g. of chlorine succumbs to the influence of silver, dissolving only 40.3 per cent. of the gold.

The data show that a supersaturated solution of clorine in water acts more strongly than one that is merely saturated, that a supersaturated solution can extract a satisfactory percentage of gold from a gold-silver alloy containing as much as 20 per cent. of silver, and that with the decrease of chlorine below a certain amount (2.1 g.) and the increase of silver above 10 per cent., the extraction of gold falls off quickly.

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[17] *Engineering and Mining Journal*, Oct. 29, 1887, p. 309.

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The results of bromination in Fig. 3 show that bromine also is an efficient solvent for gold, giving extractions of 99.8, 98.2, 96.6, 93.2 and 92.2 per cent. with solutions of 8.97, 5.98, 4.48, 2.90 and 2.60 g. of bromine in 150 c.c. of water; when the bromine present sinks to 2.1 g. the yield in gold falls quickly to 34.2 per cent., and then diminishes more gradually.

Observations made with the flasks during the tests are recorded in Table V.

TABLE V.—*Observations During Bromination-Test.*

Bromine in 150 c.c. of Water.	Extraction of Gold.	Color of Solution.	Odor of Solution.	Pressure of Solution.
Grams.	Per Cent.			
8.97	99.2	Dark-red.	Strong.	None.
5.98	98.2	Dark-red.	Strong.	None.
4.48	96.6	Dark-red.	Strong.	None.
2.99	93.2	Dark-red.	Slight.	None.
2.6	92.2	Dark-red.	None.	None.
2.1	34.2	Light-red.	None.	None.
1.5	23.8	Pale-red to yellow.	None.	None.

The experiments of Dancer[18] show that at 15° C., ordinary temperature, saturated bromine water contains 3.22 per cent. of bromine; a saturated solution of 150 c.c. of water contains 4.839 g. of bromine. The tests show that a solution with as little as 2.6 g. of bromine dissolves more than 90 per cent. of the gold when silver is absent.

The presence of 20 per cent. of silver in the gold does not make more than 1 per cent. difference in the extraction in the first four tests with 8.97, 5.98, 4.48 and 2.99 g. of bromine; the fall begins with 2.6 g. of bromine which dissolve 77.6 per cent. of gold, while in the absence of silver the same amount of bromine gives an extraction of 92.2 per cent.

With an alloy of 7 parts gold and 3 parts silver, only the solution with 8.97 g. of bromine holds its own, giving a yield of 97.7 per cent.; the other solutions show diminishing extractions corresponding with the smaller amounts of bromine present.

With an alloy of 6 parts of gold and 4 parts of silver, the strongest solutions are similar to the weaker ones in their low percentages of extraction.

[18] Comey, *op. cit.*

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The data show that 150 c.c. water containing only 1.93 g. of bromine give a satisfactory extraction (92.8 per cent.) of gold from an alloy of gold with as much as 20 per cent. of silver, and that the extraction figure is raised only 4.7 per cent. by doubling the amount of bromine.

The comparison of the dissolving-effects of chlorine and bromine upon pure gold and upon gold alloyed with different amounts of silver can be seen in Fig. 3, but is more clearly illustrated in Fig. 4.

Comparing curve I. (chlorine and pure gold) with curve II. (bromine and pure gold), it is seen that with concentrated solutions (2.8 g. of Cl equivalent to 6.0 g. of Br), chlorine is a slightly stronger solvent than bromine; as the solutions become less concentrated the dissolving power of chlorine falls more quickly than that of bromine, until at 1.58 g. chlorine (equivalent to 3.4 g. of bromine) the power of the two is the same. With further dilution to 1.20 g. of chlorine, bromine is a much better solvent than chlorine; when the dilution is carried very far, the percentages of extraction by the two solvents come again nearer together, but bromine always remains the more efficient of the two.

Curve III. shows that the extraction of gold from an alloy of gold 9 parts and silver 1 part by means of chlorine is but little affected by the presence of 10 per cent. of silver; that this is still less the case with bromine can be inferred from the bromination curves in Fig. 3, where pure gold and gold with 20 per cent. of silver are connected by straight lines.

Coming to the alloy of 8 parts of gold and 2 parts of silver, the harm done by the silver on the extraction with chlorine (curve IV.) is marked, while with bromine it has very little effect. While with 2.8 g. of chlorine the extraction of gold is diminished only 2.1 per cent., it falls quickly and regularly with dilution of the solvent to 22.1 per cent. until 1.3 g. has been reached, and then falls suddenly to an insignificant figure. With bromine, dilution from 6 to 3 g. hardly affects the result, and a further addition of water only very little. Thus bromine again is the better solvent in the presence of silver.

Curves VI. and VII., representing the treatment of an alloy of 7 parts of gold and 3 parts of silver, with chlorine and bromine, show how seriously the extraction of gold is affected by

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the presence of such large proportions of silver, but the extractions of bromine are always better than those of the equivalent quantities of chlorine.

Curve VIII., standing for the alloy of gold 6, and silver 4, finally shows how even bromine loses its dissolving power when it meets such large proportions of silver.

The curves bring out the additional interesting fact that changes in the degree of concentration of chlorine, as well as of bromine, do not make a very decided difference in the percentage of extraction with pure gold or gold with 10 per cent. of silver as long as with 150 c.c. of water, 1.4 g. of chlorine, equivalent to 3 g. of bromine, is not overstepped as the lower limit. With gold containing more than 10 per cent. of silver, a slight decrease in the concentration makes a very decided difference in the extraction until 1.2 g. is reached with chlorine, and 2.1 g. with bromine. If the solutions are further diluted, the effect of an increase in silver is not so very marked.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Labor-Saving Appliances in the Works-Laboratory.

BY EDWARD KELLER, BALTIMORE, MD.

(Annual Meeting, February, 1905.)

THE present ruling principle in shop and factory, induced by conditions of keen competition, is to do the greatest amount of work in the shortest time, or in other words, to secure the greatest output at least cost,—a condition which generally necessitated the substitution of machinery for manual labor.

The chemical laboratory in our industrial establishments has remained a factor of minor importance, and, for this reason, not only has progress been permitted to lag, but the methods of analytical work have been carried on along individual traditional lines. Some works-laboratories are equipped with such conveniences as Gooch filters, suction-pumps, compressed air, shaking-machines, centrifugal machines, crushers, etc., yet there is much room for improvement, particularly in laboratories of metal smelting and refining plants, in which long series of filtrations and stirrings are performed singly by hand in a slow and laborious manner. There are also many other instances in which collective handling appears rational. As a rule the chemist wastes much time by such manual labor, which, in a modern establishment and under up-to-date management, could be more advantageously applied to useful chemical work; and such higher chemical work is gradually and surely becoming recognized as an essential extension of old-time methods of assaying. While it may be said that in some laboratories, boys and not chemists do the manual labor, it should be borne in mind that all chemical work requires care and accuracy and in these respects boys are naturally a source of concern to the responsible chemist. Moreover, the cheapness of child-labor is largely offset by the increase in the expense-account of laboratory materials, caused chiefly by breakages resulting from carelessness.

Having been authorized by the Anaconda Copper Mining

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Co. to build a new laboratory in Baltimore, Md., I decided to make its equipment superior to any that I had seen in this country and abroad. As in many others, the bulk of the work done in the works-laboratory of our company consists in the routine of a few methods applied to many determinations. At Baltimore the value is determined of the copper, silver and gold contained in the crude material that is handled in the works. This analytical work is commonly called assaying,—an old and useful art which, however, is too often lightly considered by those having an excessive knowledge of chemistry and metallurgy.

General Arrangement of Laboratory.

Concerning the general arrangement of the laboratory, but little need be said. Mr. Charles H. White, in a paper entitled The Equipment of a Laboratory for Metallurgical Chemistry in a Technical School,[1] which was read at the Atlantic City Meeting of the Institute, February, 1904, has pointed out the desirability of arranging a laboratory in a technical school in such a manner that the greatest amount of work can be done by each student. This systematic arrangement is equally desirable in the works-laboratory, and the one under consideration has been so designed that the number of movements required from the beginning to the end of every operation has been reduced to a minimum.

The hygienic arrangements of the room should also be given proper attention, especially the ventilation, which as a rule is only made satisfactory by the proper construction of the hoods. At our new works-laboratory the two-compartment hood, built of brick, is plastered inside, has a tile floor, the openings into the laboratory being supplied with vertical sliding glass doors. The stack, 21 ft. high, is directly over the slate roof of the hood, which tapers from two sides toward the openings of the stack. The cross-section of the openings from the hood to the stack equals that of the free space in the stack. The hood is built against the wall of the building, and parallel to this wall is a second one, leaving a free space of 4 in. between them which communicates with the hood by a number of 2- by 4-in. holes,

[1] *Trans.*, XXXV., 117 (in press).

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along the floor, and with the stack by a larger opening at the highest point of the hood. This construction gives a good draught along the floor, as well as at the apex of the hood, and, regardless of the amount of copper that is being dissolved with nitric acid, there is never any noxious odor of "red fumes" in the work-room of our laboratory. A plan and part section of the new laboratory of the Anaconda Copper Mining Co. at Baltimore, Md., are given in Figs. 1 and 2 respectively.

With regard to cleanliness of work in the laboratory, my first teacher in practical chemistry, Hermann von Fehling, used to tell his students, that "*Der Chemiker muss im Frack-Auzug arbeiten koennen.*" This remark made a lasting impression upon me, and I have always disliked the sight of black or yellow hands, or of work-clothes having many spots and holes. It is not given to everyone, however, to be as tidy as Fehling was, but it is possible to aid natural defects by artificial means.

Conveniences for Handling Beakers and Acids.

As already stated, assaying forms the bulk of the work at the Baltimore laboratory. Scores of silver-determinations and hundreds of scorifications and cupellations for gold are often made. Silver, and sometimes gold, is determined in the metallic copper-material by what is commonly known as the combination method, which consists in dissolving both copper and the contained silver in nitric acid, leaving the gold as a metallic residue. The silver is re-precipitated in the form of chloride and, with the gold, is separated from the copper solution by filtration. The incineration of the filter, scorification with metallic lead, and cupellation with a subsequent parting of the two precious metals, completes the assay. We have simplified this operation first by the introduction of metallic trays to hold the No. 5 Griffin beakers, thus avoiding handling them either singly or in pairs. The trays, shown in Fig. 3, hold nine beakers, and as soon as the copper has been dissolved the tray may be placed directly on the fire in order to expel the nitrous fumes by boiling the solutions. The acid used to dissolve the copper is kept in a 5-gal. bottle, having a glass cock, or spigot, with a 0.25-in. orifice, through which the outflow is very rapid. From this bottle the acid is tapped into a specially designed measuring-cylinder shown in Fig. 4, provided with a stop-cock similar to that

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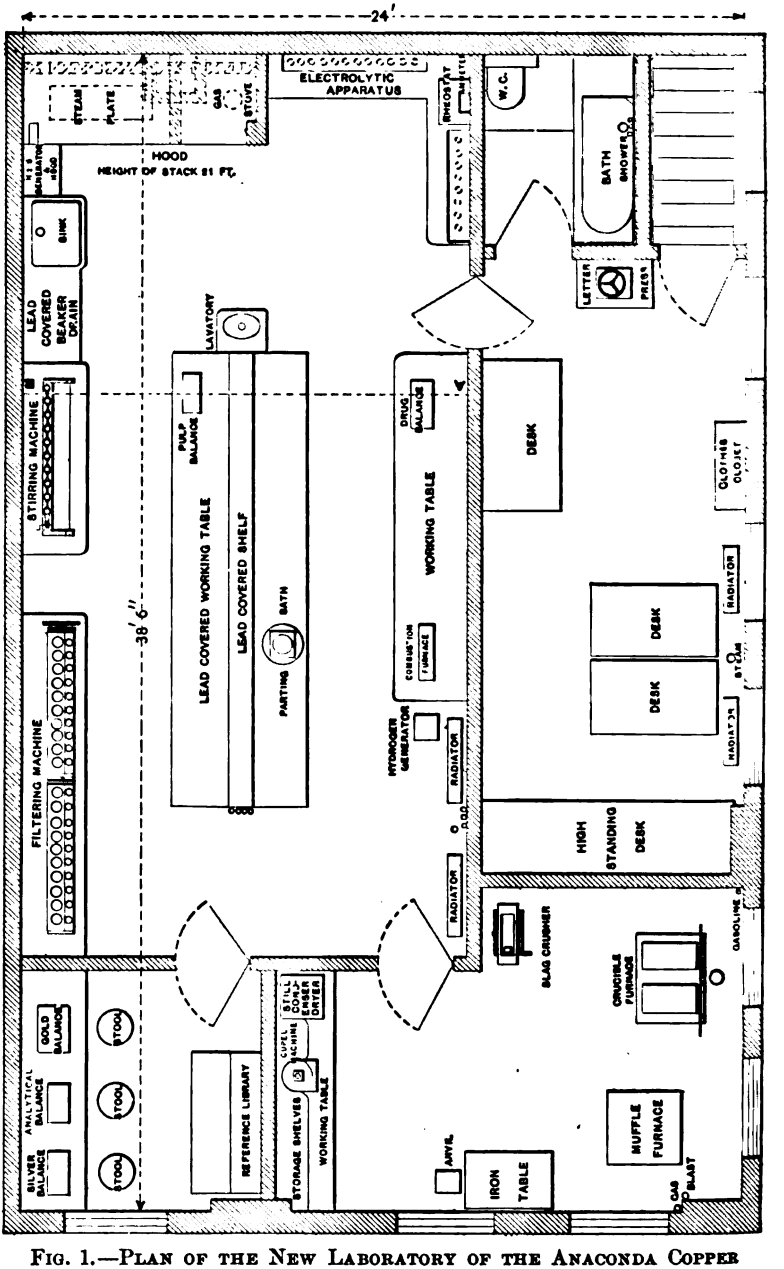

FIG. 1.—PLAN OF THE NEW LABORATORY OF THE ANACONDA COPPER MINING CO. AT BALTIMORE, MD.

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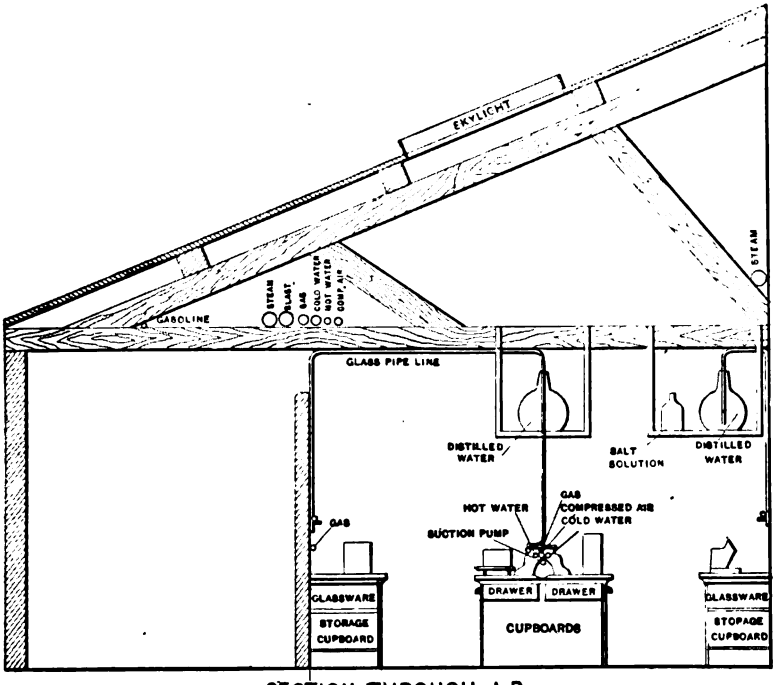
of the bottle, which allows the acid to be tapped into the beakers, without dripping, in a neater and cleaner manner than can be done by pouring from a bottle, beaker or cylinder.

After the copper solutions have become cold and the proper quantity of sodium chloride requisite to precipitate all the silver has been added, it is absolutely necessary to stir the mixture in order that the reaction shall take place throughout, and that, after adequate settling, none of the silver chloride will run

FIG. 2.



SECTION THROUGH A B

SECTION THROUGH PART OF THE NEW LABORATORY OF THE ANACONDA COPPER MINING CO. AT BALTIMORE, MD.

through the paper during the subsequent filtration. The stirring of a large number of solutions by hand occupies much time and is very tedious, but omitted, the determination becomes faulty. Figs. 5, 6 and 7 illustrate a stirring-machine which obviates all the difficulties incident to the manual operation of stirring and, by its use ten solutions can be stirred in the same length of time occupied in stirring one by hand, and there is no splashing of solution or breakage of beakers.

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Fig. 5 shows the machine ready for use; Fig. 6 shows the position after the stirring-rods have been lifted from the solutions and are ready to be rinsed; and Fig. 7, the position assumed after the rods have been rubbed off, the disks holding them and forming a cover to the beakers, now being ready to be washed. This latter operation, however, is generally superfluous, for very seldom does any of the solution reach the covers.

The characteristics of this stirring-machine are the three-based feet, the convenient driving-gear on the right which admits of the ready application of mechanical power; the slack take-up for the belt on the left; and the rubber disks which hold the rods and form covers to the beakers. The action of the machine is made universal by having several holes in each disk so that the position of the rods will conform to any size of beaker. The separate construction of both beaker-stand and stirring-stand permits a ready change of a set of beakers and allows the rods to remain a permanent part of the whole. Descriptions of stirring-machines are given in some catalogues of manufacturers of chemical apparatus, but they seem to lack the essentials necessary for successful manipulation.

Filtering Apparatus.

The construction and operation of our filtering or decanting apparatus are illustrated in Figs. 8, 9 and 10; and by its use 20 filtrations can be performed with perfect ease, and without the least danger of loss by splashing or breakage. The beaker-rack is tilted by means of a hand-wheel on the right, retrograde motion being prevented by a ratchet. The point of rotation of the whole series of beakers lies some distance from their lips, about at the end of the glass rods that guide the stream of liquid to a definite point in the filter, an arrangement which is essential for steady pouring. The lifting of the load is aided by a counterpoise on the left. By the use of this apparatus the filtering can be done leisurely with one hand, so steadily that the precipitate remains undisturbed until all the clear liquid is poured off, and the time of the whole operation is greatly shortened. Figs. 8 and 9 show the machine as originally constructed, with a locking-device for each beaker and rod, consisting of an arm pressed downward by a spring, and free to rotate vertically within an angle sufficient to clamp, as

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well as to allow it to sweep horizontally over the rod and beaker, the horizontal rotation also being entirely free. Fig. 10 presents a locking-device on the same machine by means of which ten beakers and rods are locked simultaneously. It is a matter of taste which arrangement is the more satisfactory. The beaker-rack and the filter-stand are adjustable to several sizes of beakers and funnels, and the machine can be made for any number of beakers. Light wooden trays are provided to carry the beakers, in sets of ten, to or from the apparatus.

Table I., containing the results of a test recorded by one of my assistants, shows the difference in time between the new method of mechanical filtration and the old method by hand. The quantity of solution contained in each beaker varied between 275 and 300 c.c.

TABLE I.—*Comparison of Results of Machine Filtration vs. Hand Filtration.*

Machine Filtration.			Hand Filtration.		
		Time Consumed. Minutes.			Time Consumed. Minutes.
10.21 a.m.	Started.		11.13 a.m.	Started.	
10.22 a.m.	Placed rods	1	11.14 a.m.	Placed rods	1
10.30 a.m.	Finished decanting	8	11.38 a.m.	Finished decanting	24
10.41 a.m.	Washed beakers and rods	11	11.58 a.m.	Washed beakers and rods	20
10.45 a.m.	Washed filters	4	12.04 p.m.	Washed filters	6
10.55 a.m.	Rubbed out beakers	10	12.13 p.m.	Rubbed out beakers	9
10.59 a.m.	Folded filters and placed in scorifiers	4	12.17 p.m.	Folded filters and placed in scorifiers	4
Total		38	Total		64

Difference in favor of machine, 26 minutes.

I have decanted ten beakers, each containing from 275 to 300 c.c. of solution in 4 min. 30 sec., the solutions being run through a double S. & S., No. 597, 12.5 cm. filter.

In connection with the work as just described we have entirely dispensed with the well-known wash-bottle, on account of the unnecessary physical strain involved in its use, as well as its unsanitary character in many instances. The distilled water contained in carboys placed about 4.5 ft. above the floor-level is siphoned into a system of glass-tubing, extending throughout the laboratory wherever the water may be needed. This glass-tubing is provided with numerous T connections to

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which are attached rubber-tubes, having pinch-cocks and glass nozzles at their ends, which permit the direction of a stream of water to any desired point. The water in the overhead carboys is replenished by forcing a new supply from another carboy in which it has been condensed, placed on a carriage on the floor, up through a glass siphon by means of compressed air.

When the filtration is complete and the precipitates have been washed out of the beakers, the latter must be rubbed out with paper in order to be sure that neither silver chloride nor gold is retained on the walls of the vessels, perhaps by a little oil often contained in the samples, or by the drying and hardening of small particles of chloride above the surface of the solution. In order to facilitate this cleansing, we have designed a machine in which the beakers are rotated. This machine is on rollers, and runs on rails fastened to the table in front of the filtering-apparatus, and may be locked at any desired point. The rubbing is done by means of a "policeman," Fig. 11, which consists of a rod having a cork fastened at one end, over which is clamped a piece of filter-paper, held in place by a conically-cut ring. The paper thus fastened is run along the bottom and side of the rotating beaker. For each beaker a fresh piece of paper is quickly clamped to the cork.

New Assay-Furnace Tools.

The implements used in performing the furnace-work in assaying are shown in Fig. 12. A and B are the traditional tongs, universally used for handling singly the scorifiers and the cupels. The former has been entirely replaced with a fork, *C*, with which a set of 20 scorifiers can be handled at one time. In silver-assaying each set of scorifiers is placed in the muffle twice and taken out twice; first put in the muffle for the incineration of the filters, then taken out for the addition of the test-lead, then returned for scorification, and finally taken out for pouring the slag and molten lead into the molds. By the use of the fork, which works perfectly if the muffle be properly supported so that it will not sag to any marked extent, 60 handlings are reduced to three. Four scorifiers constituting a longitudinal row in the muffle are poured at one time by means of a pair of tongs, *D*, and with a little practice the pouring is made just as easy as with a single scorifier. It is neces-

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sary only that the mold correspond to the arrangement of the scorifiers and that the pockets are shallow. The cupels are placed into, or taken from, the muffle in sets of one or more rows, by means of the tools, *E*, *F* and *G*., an idea which, I believe, was first put into practice by my brother, Richard Keller, of Durango, Colo. *E* and *G* are sharp-edged shovels, the latter having upturned sides. *F* is a rabel, with which the cupels are raked onto the shovel, removed therefrom to the place where they are to be deposited by placing it behind them and withdrawing the shovel. The tool, *H* and *I*, is entirely new, and by its use one or more rows of cupels in the muffle may be charged with the lead-buttons from the scorifiers. Fig. 13 shows the idea of the device more clearly. It comprises a top sliding-plate with openings corresponding exactly to the position of the cupels. The openings in the lower plate correspond exactly with those of the upper one; the plate, however, rests on two adjacent sides extended downward at right-angles to the plate and to each other, thus forming two closed sides of the instrument; one at the front, and the other at the right-hand side. The height of these sides is such that, when resting on the bottom of the muffle, the bottom plate will be some distance above the cupels and, by a slight pull forward and a push to the left with the handle of the instrument, the set of cupels will be perfectly aligned in both directions, and the apertures in the lower plate will exactly cover the tops of the cupels. The lead-buttons are placed in the apertures of the upper plate and rest on the lower plate before introducing the instrument into the furnace, and when it is placed over the cupels, which have been properly aligned in the muffle, the upper plate is pushed forward to a stop-point, bringing the apertures of the two plates to register, thus causing the lead-buttons to drop down into the cupels. The handle of the upper plate runs through guides fixed to the handle of the lower plate; both handles are connected by a spring, which acts as a brake when the upper plate is pushed forward to drop the buttons, and also serves to bring it back into its original position, in which the buttons cannot drop through the apertures in the lower plate.

Charles Tookey[2] first recommended the use of hydrochloric

[2] *Journal of the Chemical Society* (London), vol. xxiii., p. 366 (1870).

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acid (1 HCl.aq, 2 H_2O) instead of a brush for cleaning the buttons, and for this purpose a small silver dish and tray having perforated pockets give excellent satisfaction. By the use of this device 50 or more beads at a time can be treated, washed and dried without transfer.

Fig. 14 shows a very handy parting-bath; which though old in principle has not been in general use. The vessel is a constant-level water-bath and the tray an ordinary test-tube holder. The silver beads to be parted are dropped into the test-tubes, and the latter filled with dilute nitric acid of a strength of one of acid (sp. gr. 1.42) to 9 of water. The water in the bath is first brought to the boiling-point before the tray with its contents is set into it. Treated in this way, the gold almost invariably remains in the form of a small coherent bead, even from an alloy as low as 1 part of gold to 500 of silver.

I have recognized that the system of handling everything in sets was incomplete as long as I was unable to take a whole set (20) of scorifiers from the muffle and pour their contents simultaneously into the molds. Recently a tool for that purpose, shown in Figs. 15 and 16, has been perfected. It is composed of quintuple tongs, corresponding to the five longitudinal rows of scorifiers in the muffle. The lower part of each pair of the tongs consists of a fork, on which the scorifiers rest, and one of whose prongs is rectilinearly extended through two bearings in a frame, and held in position by collars. This extension is free to revolve in the bearings, and it is the axis of rotation of the tongs. To each of them is attached, at right angle, a lever, extending upward at an angle of 45°, and all the levers are connected by slotted joints to a cross-rod. Therefore, if by means of a crank, fastened to the end of one of the extended prongs, one of the forks is turned and the scorifiers tilted to the desired angle, the others perform the same rotation. The center of gravity of the scorifiers lies to one side of the rotation-point, and they would therefore, on being lifted, tilt in that direction; this, however, being prevented by the cross-bar resting against a post at that end of the frame toward which the inclination tends.

The scorifiers are clutched by the upper prong of the tongs, which is fastened to a spring on a post of the fork below, and which is free to move in a vertical plane; the pivotal point

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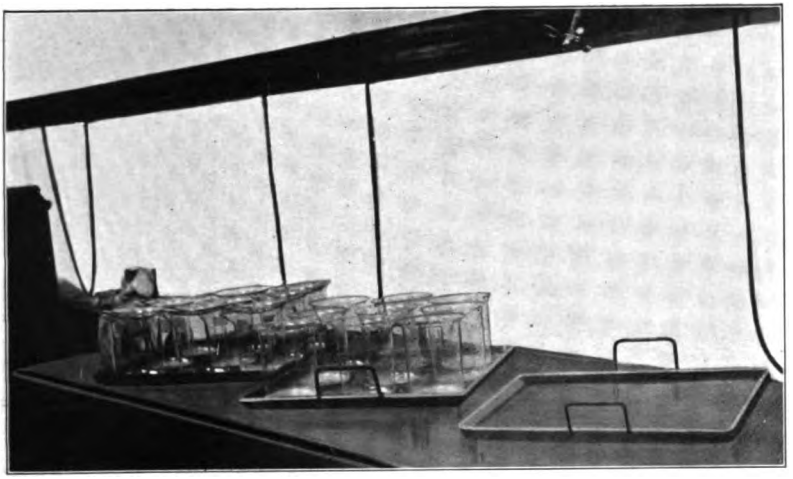

FIG. 3.—METALLIC TRAYS HOLDING NINE NO. 5 GRIFFIN BEAKERS.

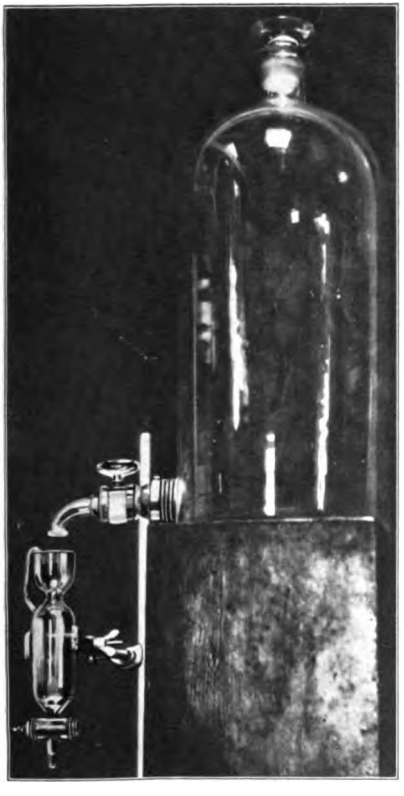

FIG. 4.—CYLINDER FOR MEASURING ACID.

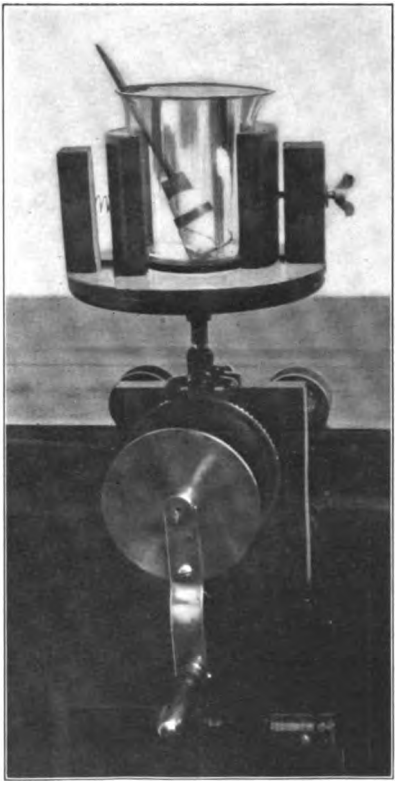

FIG. 11.—THE "POLICEMAN" IN BEAKER IN ROTATING MACHINE.

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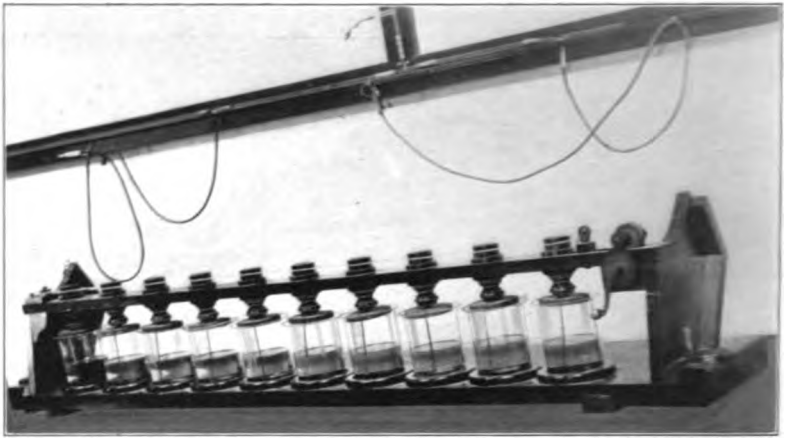

FIG. 5.—In position ready for use.

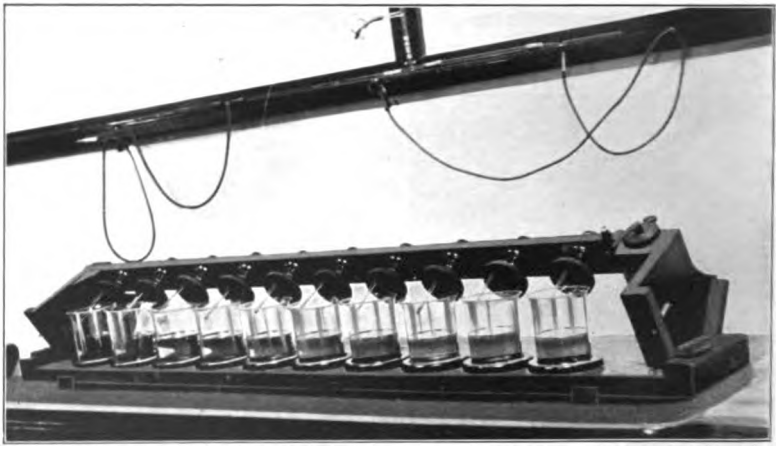

FIG. 6.—In position ready for rinsing, after lifting out the stirring-rods.

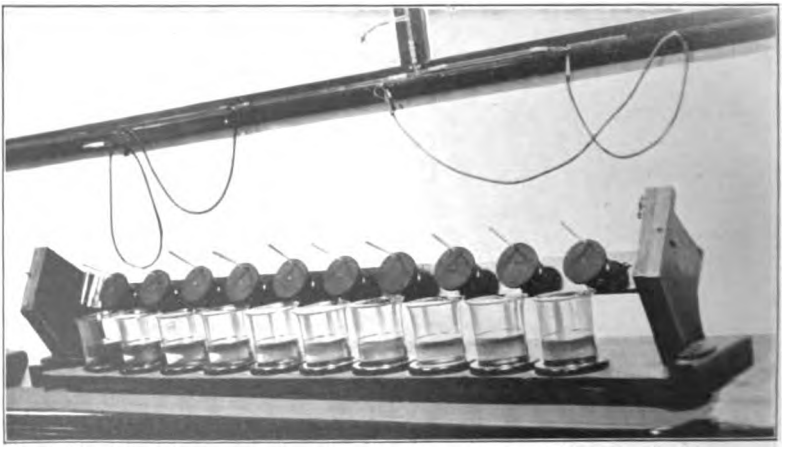

FIG. 7.—In position ready for washing off the disks.

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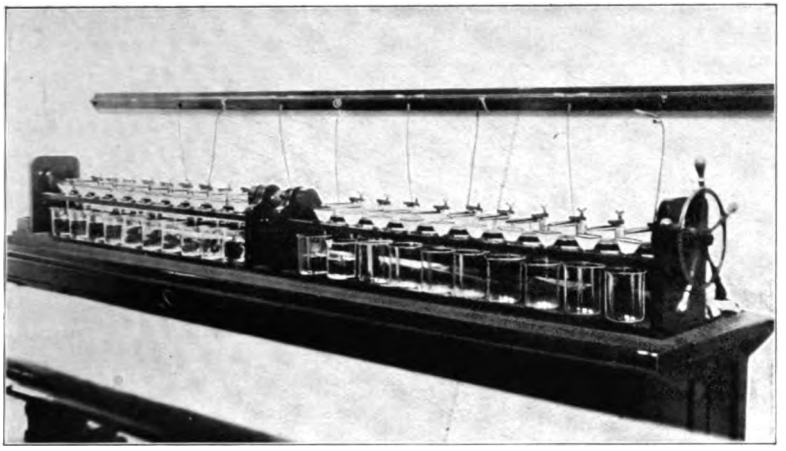

FIG. 8.—Original machine, ready to begin Filtration or Decantation.

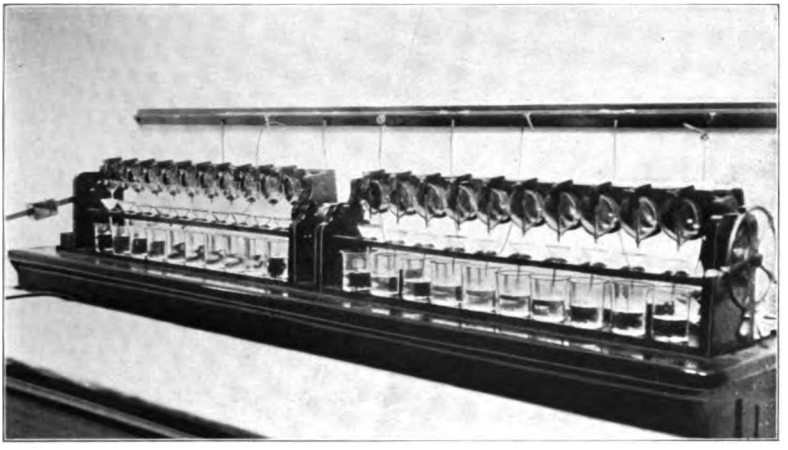

Fig. 9.—Original machine, position at end of filtration.

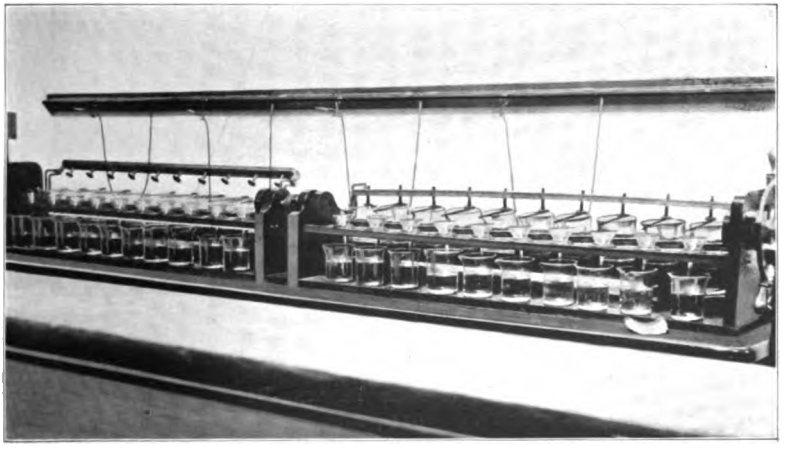

FIG. 10.—Original machine with new locking-device.

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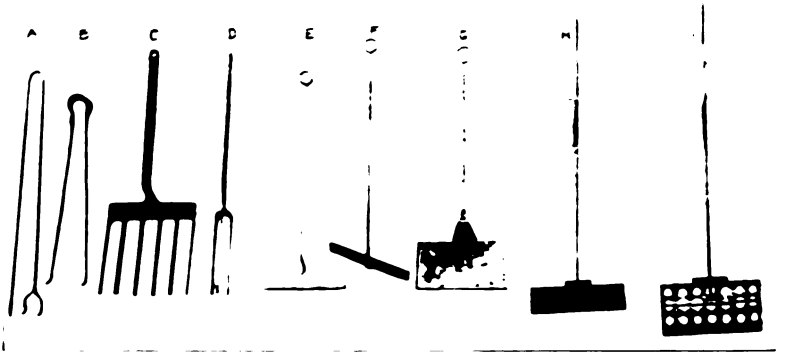

FIG. 12.—FURNACE TOOLS.

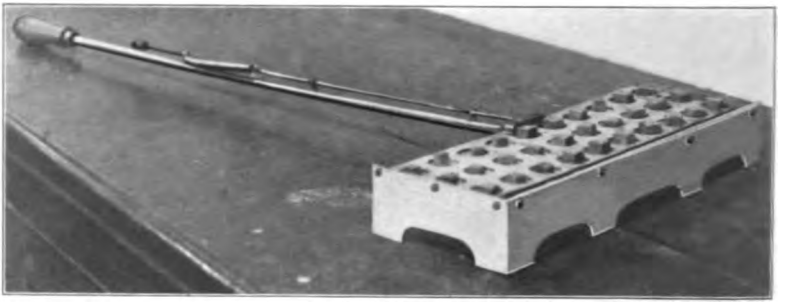

FIG. 13.—DEVICE FOR CHARGING SCORIFIER BUTTONS INTO CUPELS.

FIG. 14.—CONVENIENT PARTING-BATH.

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lying over the spring and post. By bringing pressure on the extended ends of these clutch-bars behind the pivot, their other end will rise from above the scorifiers, and thus release these, or permit the placing of them onto the tongs. The pressure exerted on the rear ends of the clutches is accomplished by means of a cross-bar fastened to a spring-bar, which is itself

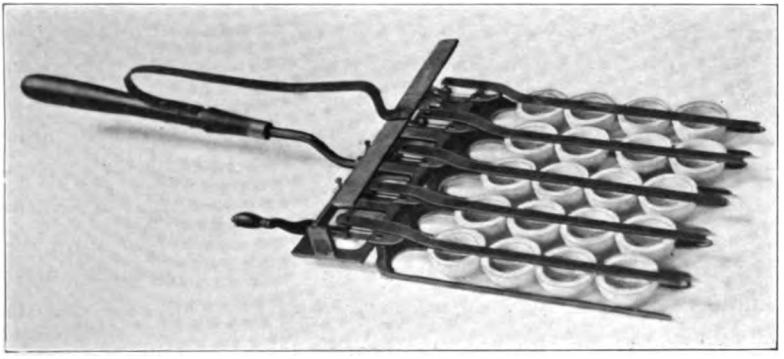

FIG. 15.—MULTIPLE TONGS; SCORIFIERS IN POSITION FOR REMOVAL FROM FURNACE.

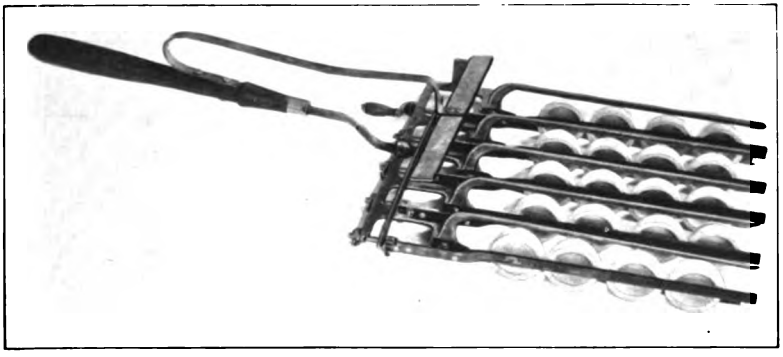

FIG. 16.—MULTIPLE TONGS; SCORIFIERS TIPPED FOR POURING CONTENTS.

riveted to the handle of the instrument; all of which may be plainly discerned in the illustrations. In pouring the contents of the scorifiers, the frame of the tool rests on the edge of the mold, leaving the tongs free to turn.

The introduction of the new system of manipulation described in this paper has resulted in economy in several ways. Much labor has been saved; breakage of expensive glass-ware

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has been very largely eliminated; and the time of the furnace-work, and, consequently, the consumption of gas, have been much reduced. Furthermore, the gain has been a moral one, and work formerly regarded as tedious has become more of a pleasure; especially has the sojourn in the furnace-room during the hot summer months been rendered cooler by being greatly shortened.

The appliances described in this paper may eliminate the laboratory boy, but if not, they will make him more reliable. They also increase enormously the quantity of chemical work that one man can do in a day. In the small domain of our laboratory the change from the old systems to the new is considered to bear about the same relation as the change from ancient horse-car to the modern rapid transit.

In conclusion, I take pleasure in acknowledging the efficient collaboration of my assistants, Mr. Albert Ferrell and Mr. K. W. McComas, the former having shown himself particularly useful as a skilled mechanic in the construction of various improvements.

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SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Concrete in Mining and Metallurgical Engineering.

A Reply to the Discussion by Mr. Francis T. Havard of the Paper by Mr. Henry W. Edwards, which was read at the Atlantic City Meeting, February, 1904.

(Annual Meeting, February, 1905.)

MR. HENRY W. EDWARDS, Grand Junction, Colo. (communication to the Secretary):*

Effect of Heat.—It should be remembered that the coefficient of expansion of concrete is nearly that of iron; therefore in constructing flues the design must take account of this important detail. The cracking of flues, mentioned by Mr. Havard as having occurred at a temperature of 100° C. and above, were possibly caused by neglecting to appreciate the extent of the expansion and contraction of the concrete.

Effect of Atmospheric Moisture.—The moisture present in the atmosphere is exceedingly injurious to flues of any material, and those made of concrete are no more susceptible than of brick. My experience with flues of the Monier construction, having walls from 1.5 to 2.5 in. thick plastered upon a net-work of iron rods, was unsatisfactory from the standpoint of durability.

* Received October 25, 1904.

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No. 3. MAY. 1905.

Bi-Monthly Bulletin

OF THE

American Institute of Mining Engineers.

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At S.-W. Cor. Seventh and Cherry Sts.

PHILADELPHIA, PA.

EDITORIAL OFFICE, AT 99 JOHN STREET, NEW YORK, N. Y.

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SECTION I.

INSTITUTE ANNOUNCEMENTS.

This section contains announcements of general interest to the members of the Institute but not always of sufficient permanent value to warrant republication in the volumes of the *Transactions*.

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(Organized in 1871, and Incorporated in 1905.)

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For the year ending February, 1906.

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BI-MONTHLY BULLETIN.

For the convenience of persons who desire to file, or otherwise use separately, the technical papers in Section II. of the Bulletin, each of these papers has been paged and wired by itself; the whole collection being held together by a single, heavy wire, upon the removal of which it will fall apart into individual pamphlets, substantially like those formerly issued.

A small stock of separate pamphlets, duplicating the technical papers given in Section II. of this Bulletin, is reserved for those who desire extra copies of any single paper.

All communications concerning the contents of this Bulletin should be addressed to R. W. Raymond, Secretary, 99 John St., New York City (P. O. Box 223; Telephone number 5477 John).

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TRANSACTIONS.

Volume XXXV. of the *Transactions* will be ready for distribution in June. The Secretary has usually attempted to issue the volume containing the papers of each year in July of the following year. But in consequence of the issue in 1903 of the extra "Mexican Volume" (Volume XXXII.), as well as Volume XXXI., the work was so much delayed that Volume XXXIII. was eight months, and Volume XXXIV. four months, behind the usual date. This year, by hard work (especially on the part of Dr. Joseph Struthers, Assistant Editor of the *Transactions*), a period has been gained, covering not only those four months but also an additional one, so that Volume XXXV. will be distributed earlier than any of its predecessors for many years past.

When the book leaves the hands of the binder in Philadelphia, it will be sent by express to every member who has paid the dues of 1905. This shipment of several thousand volumes at one time is, like all operations at wholesale, comparatively easy and cheap. The subsequent shipment, from the New York office, of a single copy to each member, upon his payment of arrears, is proportionately much more troublesome and expensive. It does not seem fair that such extra pains and outlay should be incurred for the benefit of those who have neglected to meet promptly their own obligations as members. Notice is therefore hereby given, that the Council reserves the right, in such cases (and in the absence of satisfactory reasons to the contrary), to deliver the volume at the cost of the member to whom it is sent.

Volume XXXV. will comprise about 1135 pages, which is a little more than the desirable maximum for secure binding and convenient use, and especially for transmission to foreign countries, under the rules of the International Postal Union. Since the experience of the last three years has demonstrated once more that the Institute cannot issue more than one regular volume of *Transactions* a year without seriously impairing the operations of its publishing department, it is imperatively necessary to condense, as far as practicable, the contents of

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the annual volume. For this purpose, it has been decided to omit (whenever required, to permit the publication of valuable original papers and discussions) some of the material heretofore included in the annual volume. In the case of Volume XXXV., the following items have been thus omitted:

1. The detailed account of the excursions connected with the Lake Superior meeting of September, 1904.

2. The obituary paragraphs of the Annual Report of the Council of February, 1904.

3. The following papers:

Acid Open-Hearth Manipulation, by Andrew McWilliam and William Hatfield;* also Discussion by J. J. Morgan, E. H. Saniter, and Reply by Messrs. McWilliam and Hatfield.†

Comparison of Methods for the Determination of Carbon and Phosphorus in Steel, by Baron Jüptner von Jonstorff (Austria), Andrew A. Blair (United States), Gunnar Dillner (Sweden), and J. E. Stead (England);* also Discussion by R. Hamilton, E. H. Saniter, and Reply by Mr. Stead.†

The Development and Use of High-Grade Tool-Steel, by J. M. Gledhill;* also Discussion by E. Windsor Richards, H. M. Howe, Tom Westgarth, H. H. Campbell, John A. Mathews, A. S. Pye-Smith, J. W. Richards, Oberlin Smith, W. Carter, H. C. H. Carpenter, John Little and F. M. Osborn.†

All of the above material has been published in full in the *Bi-Monthly Bulletin;* and the papers named have been published in the *Journal of the Iron and Steel Institute.* With regard to the obituary paragraphs, it should be said, that additional data are received from time to time (sometimes long after the date of the Report of the Council); and that it is intended hereafter to publish such notices in the *Bi-Monthly Bulletin*, as soon as they have been perfected, and to prepare at a future time a special volume, containing the notices for a series of years, with a suitable index for reference.

---

* Published in *Bi-Monthly Bulletin*, No. 2, March, 1905.

† Published in *Bi-Monthly Bulletin*, No. 3, May, 1905.

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LIBRARY.

The set of the *Journal of the Iron and Steel Institute* has been completed through the generosity of Mr. Arthur Thacher, who has presented a perfect copy of the volume for the year 1871.

The following volumes are needed to complete the sets now in the library.

Berg- und Hüttenmännische Zeitung, volumes i. to xxi. inclusive.

Stahl und Eisen, Nos. 1, 2, 3 and 4 of vol. i.; Nos. 8, 9, 11 and 12 of vol. ii., and indexes of vols. i., ii. and iii.

North of England Institute of Mining and Mechanical Engineers, vols. xxii., xxiii., xxiv., xxv. and xxvi.

Members of the Institute and others interested in its work are asked to coöperate with the library in securing the lacking parts or volumes of the above-named publications, also to present duplicate copies of works on subjects of interest to Mining Engineers.

Accessions.

February 16, to April 15, 1905.

FROM AGUILAR Y SANTILLAN, RAFAEL.

SOCIEDAD GUANAJUATENSE DE INGENIEROS. *Boletin*, vol. i., Nos. 2–10; vol. ii., Nos. 1–2, 4–5; vol. iii., Nos. 1–3; 8vo, Guanajuato, 1888–'93.

——— *Datos para la Historia del Colegio de Mineria*, 496 p., 8vo, Mexico, 1894.

——— *Memorandum de Fisica Elemental Arreglado por Jesus Gasca*, 27 p., 8vo, Guanajuato, 1892.

SOCIEDAD CIENTIFICA "ANTONIO ALZATE," *Memorias y revista*, vol. xxi., Nos. 1–4, 8vo, Mexico, 1904.

" AMERICAN ASSOCIATION FOR THE ADVANCEMENT OF SCIENCE. *Proceedings*, vol. liii., 8vo, Washington, 1904.

" AMERICAN INSTITUTE OF MINING ENGINEERS.

OFFICIAL HOTEL RED BOOK AND DIRECTORY COMPANY. *Hotel Red Book and Directory*, 1903, xxii + 686 p. il., 8vo, New York, 1903.

PENNSYLVANIA STATE LIBRARY. *Report of the State Librarian*, 1889–'90, 8vo, Harrisburg, 1891.

" AMERICAN MUSEUM OF NATURAL HISTORY. *Bulletin*, vols. xix.–xx., 8vo, New York, 1903–'04.

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FROM AMERICAN SOCIETY OF CIVIL ENGINEERS.

Catalogue of the Library, 700 + 4 p., 8vo, New York, 1900.

Constitution and List of Members, 1905, 8vo, New York, 1905.

Transactions, vol. liv., 8vo, New York, 1905.

" AMERICAN SOCIETY OF MECHANICAL ENGINEERS.

Contents and General Index to vols. i.–xx., 8vo, New York, 1899.

Transactions, vol. xxv., 8vo, New York, 1904.

Year Book, 26, 8vo, New York, 1905.

" AUSTRALASIAN INSTITUTE OF MINING ENGINEERS.

Transactions, vol. x., 8vo, Melbourne, 1905.

" BAUM, BERGASSESOR.

Die Verwertung des Koksofengases insbesondere seine Verwendung zum Gasmotorbetriebe, iv + 124 p. il. pl., 4to, Berlin, 1904.

" BELL, R. N.

Geology and Mineral Resources of Idaho, 21 p., 8vo, n. t. p., [1904].

" CANADIAN MINING INSTITUTE.

BREWER, W. M. *Bornite Ores of British Columbia and the Yukon Territory*, 9 p., 8vo, 1905.

BROCK, B. W. *Poplar Creek and Other Camps of the Lardeau District*, 27 p., il. map, 8vo, 1905.

CARTER, W. E. H. *The Mines of Ontario*, 54 p. il., 8vo, 1905.

CORKILL, E. T. *Notes on the Occurrences, Production and Uses of Mica*, 24 p. il. pl., 8vo, 1905.

DRESSER, J. A. *The Bed-Rock of the Gilbert River Goldfields, Quebec*, 4 p. il., 8vo, 1905.

——— *A Note on Varieties of Serpentine in Southeastern Quebec*, 3 p., 8vo, 1905.

HAMILTON, J. F. *The Relative Attraction of Some Common Minerals for Residuum Oil*, 7 p., 8vo, 1905.

HILLE, F. *A Correction in the Classification of our Gold Formation*, 9 p., 8vo, 1905.

JENNINGS, E. P. *The Goldfield District, Nevada*, 7 p. il., 8vo, 1905.

Journal, vol. vii., 8vo, Toronto, 1905.

LEONARD, R. W. *Notes on Concrete*, 7 p. il., 8vo, 1905.

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FROM CANADIAN MINING INSTITUTE (*continued*).

MICKLE, G. R. *Value of Undeveloped Mining Claims*, 7 p. pl., 8vo, 1905.

MILLER, W. G. *Undeveloped Mineral Resources of Ontario*, 20 p., 8vo, 1905.

OBALSKI, J. *A New Mining District in the North of Quebec*, 4 p., 8vo, 1905.

PARKS, W. A. *The Study of Stratigraphy*, 9 p., 8vo, 1905.

SJÖSTEDT, E. A. *A Canadian Dellwik-Fleischer Water Gas Plant*, 12 p. il., 8vo, 1905.

SPELLER, F. N. *Wrought-Pipe—Threading and Relative Durability of Steel and Iron*, 10 p. pl., 8vo, 1905.

WILLMOTT, A. B. *The Exploration of the Ontario Iron Ranges*, 13 p., 8vo, 1905.

" CANADIAN SOCIETY OF CIVIL ENGINEERS.

ARCHIBALD, E. M. *Effect of Load Factor on Cost of Power*, 14 p. il., 8vo, 1904.

DAVIDSON, S., and MCKENZIE, N. *The Mine Fire at Dominion No. 1 Colliery, Glace Bay, N. S.*, 12 p. pl., 8vo, 1904.

FORREST, B. J. *Bridge Launching*, 3 p. pl., 8vo, 1904.

SMITH, C. B. *Construction of Canadian Niagara Power Company's 100,000-H.P. Hydro-Electric Plant at Niagara Falls, Ont.*, 19 p. pl., 8vo, 1905.

" CARNEGIE LIBRARY OF PITTSBURGH.

ELLIOTT, A. M. comp. *Contemporary Biography. References to Books and Magazine Articles on Prominent Men and Women of the Time*, 171 p., 8vo, Pittsburg, 1903.

PITTSBURGH, CARNEGIE LIBRARY. *Annual Report*, vols. i.–ii., iv., v., vii., viii., 8vo, Pittsburg, 1897, 1898, 1900, 1901, 1903, 1904.

——— *Catalogue of the J. D. Bernd Department of Architecture in Carnegie Library of Pittsburgh*, 33 p., 8vo, Pittsburg, 1898.

——— *Classified Catalogue of the Carnegie Library of Pittsburgh*, pts. 2–4, 3 vols., 8vo, Pittsburg, 1903–'04.

——— *List of One Hundred Entertaining Biographies*, 19 p., 8vo, Pittsburg, 1902.

——— *List of Subject Headings for Use in Dictionary Catalogues of Children's Books*, 58 p., 8vo, Pittsburg, 1903.

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PITTSBURGH, CARNEGIE LIBRARY. *Some Information about the Carnegie Library of Pittsburgh*, 50 p., 8vo, Pittsburg, 1902.

——— *Story Telling to Children from Norse Mythology and the Nibelungenlied. References to Material on Selected Stories, together with an Annotated Reading List*, 48 p. pl., 8vo, Pittsburg, 1903.

" CARNEGIE MUSEUM, PITTSBURGH.

Prize Essay Contest, 68 p., 8vo, Pittsburg, 1904.

" CHIEF OF ENGINEERS OF THE U. S. ARMY.

U. S.—CORPS OF ENGINEERS. *Analytical and Topical Index*, 1866–1900, 3 vols., 8vo, Washington, 1902.

——— *Annual Report*, vol. v., pt. 1; vol. vi., pt. 2; vol. vii., pt. 3; vol. viii., pt. 4, 8vo, Washington, 1904.

——— ——— *Supplement*, vol. ix., 8vo, Washington, 1904.

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U. S.—ORDNANCE DEPARTMENT. *Report of the Test of Metals*, 1903, 8vo, Washington, 1904 (2 copies).

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" DEPARTMENT OF AGRICULTURE.

U. S.—BUREAU OF FORESTRY. *Bulletin*, Nos. 10, 12, 17, 28, 47, 48, 51, 54, 8vo, Washington, 1895, '98, 1900, '02, '04–05.

" DEPARTMENT OF MINES OF WESTERN AUSTRALIA.

WESTERN AUSTRALIA—DEPARTMENT OF MINES. *Report*, 1903, f°, Perth, 1904.

" DOUGLAS, JAMES.

National Geographic Magazine, vol. xvi., March and April, 1905, 8vo, Washington, 1905.

" DWIGHT, T.

AMERICAN SOCIETY FOR TESTING MATERIALS. *Proceedings*, vol. vii., 8vo, Philadelphia, 1904.

CLARK, D. K. *A Manual of Rules, Tables and Data for Mechanical Engineers*, ed. 5, xxvii + 984 p. il., 8vo, New York, 1890.

IRON AND STEEL INSTITUTE. *Journal*, vol. lxvi., 8vo, London, 1905.

SOCIETY OF BEAUX ARTS ARCHITECTS. *Constitution, By-Laws, List of Members*, 30 p., 12mo, New York.

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ARKANSAS—BUREAU OF MINES. *Biennial Report*, 1, 8vo, Little Rock, n. d.

BOOTH, W. H., and KERSHAW, J. B. C. *Smoke Prevention and Fuel Economy*, 194 p. il., 8vo, New York, 1905.

Coal Statistics, 1897–'99, 16mo, Philadelphia, 1897–'99.

DAVIS, G. E. *Handbook of Chemical Engineering*, ed. 2, 2 vols., 8vo, Manchester, 1904.

GOLDINGHAM, A. H. *Design and Construction of Oil Engines*, xviii + 255 p. il. pl., 12mo, New York, 1904.

HAMLIN, A. C. *History of Mount Mica*, 72 p., 8vo, Bangor, 1895.

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JAMES, T. P. *Handbook of Georgia*, vii + 276 p., 8vo, Atlanta, 1876.

KANSAS—INSPECTOR OF COAL MINES. *Biennial Report*, 5, 8vo, Topeka, 1892.

MICHIGAN—STATE SALT INSPECTOR. *Annual Report*, 1893, 1894, 1896, 8vo, Lansing, 1893, 1895, 1897.

MISSOURI—STATE MINE INSPECTOR. *Annual Report*, 13, 8vo, Jefferson City, 1900.

Notes and Formules de l'Ingenieur et du Constructeur Mécanicien, ed. 14, xvi + 1818 p., 16mo, Paris, 1905.

NURSEY, W. R., *Menominee Iron Range*, 150 p. il. pl., map, 4to, Milwaukee, 1891.

OHIO—CHIEF INSPECTOR OF MINES. *Annual Report*, 19, 8vo, Norwalk, 1894.

ONTARIO—BUREAU OF INDUSTRIES. *Annual Report*, 11, 8vo, Toronto, 1894.

PHILLIPS, H. J. *Gold Assaying*, xii + 138 p. il., 12mo, New York, 1904.

POOR, H. V. and H. W. *Manual of Railroads*, 8vo, New York, 1902.

POTTS. *Mining Register and Directory*, vol. iii., 8vo, North Shields, 1890.

Reports of Mining Companies:

ADVENTURE CONSOLIDATED COPPER COMPANY, 8vo, 1904.

AFRICAN CONCESSIONS SYNDICATE, 4to, 1904.

ALASKA TREADWELL GOLD MINING COMPANY, 4to, 1904.

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SECTION II.

TECHNICAL PAPERS AND DISCUSSIONS.

[The American Institute of Mining Engineers does not assume responsibility for any statement of fact or opinion advanced in its papers or discussions.]

A detailed list of the papers contained in this section is given in the Table of Contents, pages i and ii.

Comments or criticisms upon all papers given in this section, whether private corrections of typographical or other errors or communications for publication as "Discussions," or independent papers on the same or a related subject, are earnestly invited.

ERRATA.

The following corrections and additions to papers contained in this Section were received too late for incorporation in the text of said papers; but will be introduced in their proper places in the volume of the *Transactions*. These corrections are here published in order to bring them promptly to the notice of the Institute.

Page	Line	
511	24	For "which" read "this".
511	25	For "and" read "which".
517	19	For "for" read "from".
518	5	For "steam" read "pressure".
538	7	For Vs $\frac{Bd}{B}$ w = Wt read Vs $\frac{Bd}{B}$ W = Wt.
538	10 to 16 (inclusive)	omit
549	37, 38	omit
551	34, 35, 36	omit
556	15, 16	omit
599	36	For "skimming" read "smelting".
600	32	For "and" read "having been".
599	36	For "as produced in" read "as drawn from".
600	31, 32, 33	Read "Many samples of slag cleaned with varying quantities of sulphide, and subjected to various periods of settling, were sent."

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

A Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series.*

BY C. K. LEITH, MADISON, WIS.

(Lake Superior Meeting, September, 1904.)

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General Geology of the Lake Superior Iron-Bearing and Copper-Bearing Series.

The geology of the Lake Superior region is of general interest, both from an economic and a scientific standpoint. The presence of large and rich iron- and copper-deposits has made a knowledge of the geology of the region necessary for purposes of exploitation and exploration.

The ores of the region are contained in rocks of pre-Cambrian age, which for most of the country up to recent years have usually been referred to as "crystalline schists" or "crystalline complex;" and regarded principally as a metamorphosed basement-unit upon which the sedimentary rocks, beginning with the Paleozoic, were laid down. In the Lake Superior region this pre-Cambrian complex presents an unusual variety of rocks with determinable relations. It has been possible, with the large expenditures which the magnitude of the iron-mining industry warrants, to work out their stratigraphy to a larger extent than has been possible in almost any other area of pre-Cambrian rocks.

The real complexity of Lake Superior geology, the confusion of names and the multiplicity of reports covering small parts of the region, are such that interested persons, other than geologists directly engaged in the work, may well hesitate to attempt to comprehend the geology of the region as a whole. The following paper is a somewhat elementary outline of the geology of the Lake Superior region with special reference to new developments not yet covered in published reports.

Geological information concerning the region has been gathered by mining companies, and by the geological surveys of Michigan, Wisconsin, Minnesota, Ontario, Canada, and the United States. Perhaps in no other part of the country have the mining men themselves spent such large sums in purely geological work. This has made possible the publication of reports and maps by the State and Government surveys in much less time and at far less cost than would have been otherwise possible. The U. S. Geological Survey is the only organization which has covered all of the region on the United States side of the boundary, and as its reports include the information gathered by the mining companies and State organi-

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zations, special reference is made to these reports. The work of the Survey was begun in the early eighties in charge of Prof. R. D. Irving, and has been continued since 1888 by Dr. C. R. Van Hise and his assistants. The results of their work appear in a series of detailed monographs and maps on each of the principal producing ore-bearing districts.[1] A final general monograph on the Lake Superior region, accompanied by revised maps of each of the districts and a general geological map of the region, is now in preparation. The features sketched in the present paper, so far as they have not been already covered in previous monographs, will be fully described in that volume.

Geological work has been much less detailed in the Canadian portion of the Lake Superior region, largely due to the fact that the ore-bearing districts thus far discovered have not seemed to warrant such large expenditures for geological work as have been made in the United States portion of the region. The Michipicoten district is the only ore-bearing district which has been mapped and described in any considerable detail,[2]

[1] The Copper-Bearing Rocks of Lake Superior, by R. D. Irving, *Monograph No. V.*, *U. S. Geological Survey* (1883).

The Penokee Iron-Bearing Series of Northern Wisconsin and Michigan, by R. Irving and C. R. Van Hise, *Monograph XIX.*, *U. S. Geological Survey* (1892).

The Marquette Iron-Bearing District of Michigan, by C. R. Van Hise and W. S. Bayley, *Monograph XXVIII.*, *U. S. Geological Survey* (1895).

The Crystal Falls Iron-Bearing District of Michigan, by J. M. Clements and H. L. Smyth, *Monograph XXXVI.*, *U. S. Geological Survey* (1899).

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The Menominee Iron-Bearing District of Michigan, by W. S. Bayley, *Monograph XLVI.*, *U. S. Geological Survey* (1904).

See also:

Correlation Papers—Archean and Algonkian, by C. R. Van Hise, *Bulletin No. 86*, *U. S. Geological Survey* (1892).

The Iron-Ore Deposits of the Lake Superior Region, by C. R. Van Hise, *21st Annual Report*, *U. S. Geological Survey*, Pt. III., pp. 305–434.

Principles of North American Pre-Cambrian Geology, by C. R. Van Hise, *16th Annual Report*, *U. S. Geological Survey*, Pt. I., pp. 571–874.

[2] The Michipicoten Iron-Ranges, by A. P. Coleman and A. B. Willmott: *Geological Series, University of Toronto Studies*, pp. 39–151 (1902).

See also *Report of the Bureau of Mines*, Ontario, 1902, pp. 152–185.

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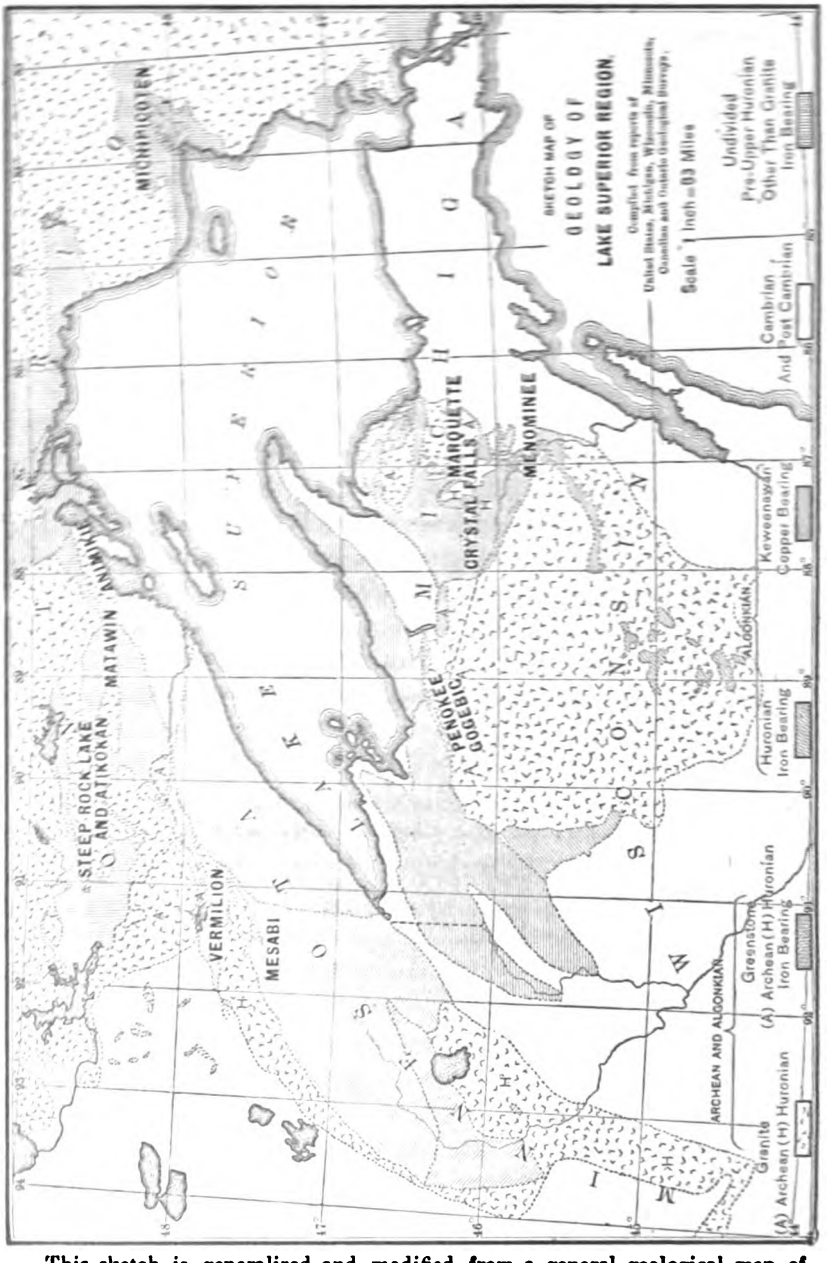

This sketch is generalized and modified from a general geological map of the Lake Superior region, on a scale of ten inches to the mile, compiled by C. R. Van Hise and C. K. Leith for publication in a final general monograph on Lake Superior geology in preparation for the U. S. Geological Survey.

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although much has been written on the general geology of the north shore.[3]

The ore-bearing districts themselves, comprising but a small portion of the Lake Superior region as a whole, are the areas in which the fullest successions of pre-Cambrian rocks are exposed; the intervening areas are less well-known, but seem to be relatively simple in lithology and structure. It has been possible to correlate with reasonable certainty the major geological units, although the geology of each of the districts has been worked out in large part independently, but because of the considerable, less well-known, areas separating the districts, it is difficult at the present time completely to unify the geology of the region.

The succession in the Lake Superior region, as given in Table I., will serve as a guide in the summary given below.

The Archean or Basement Complex.

The oldest rocks in the Lake Superior region have considerable lithological complexity. It was formerly supposed that they were largely granites and gneisses with but subordinate amounts of basic igneous rocks, but it is now known that they include large quantities of basic igneous rocks and also two or more sedimentary formations. The abundant rocks are granites, gneisses, syenites, acid porphyries, basalts, gabbros, peridotite, mica-schists, hornblende-schists, and chlorite-schists (some of the schists resulting from the alteration of the above-named rocks), jaspers, iron-ore and slates. It is not possible fully to state the stratigraphic relations of these rocks, but it is known that certain of the basalts with associated schists, known collectively by the terms greenstones and green schists, are the oldest; that associated with them and perhaps in part later and above these are the iron-formation and slate, and that intrusive into both the basal greenstones and sedimentary rocks are considerable masses of acid igneous rocks. The basic igneous rocks, commonly referred to as greenstone and green schist, with their associated slates and iron-formation, are called "Keewatin," (following Lawson and the report of the

[3] See reports of Ontario and Canadian Geological Surveys.

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TABLE I.—*Correlation of Pre-Cambrian*

		Mesabi District.	Penokee-Gogebic District.	Vermilion District.	Marquette District.
ALGONKIAN.	KEWEENAWAN. (Copper-bearing.)	Great basal gabbro and granite.	Gabbros, diabases, etc.	Great gabbro.	
	UPPER HURONIAN. (Iron-bearing.)	*Mesabi Series:* Virginia formation (upper slate). Biwabik formation (iron-bearing and productive). Pokegama formation (quartzite and quartz-slate).	*Penokee-Gogebic Series:* Tyler formation (upper slate). Ironwood formation (iron-bearing and productive). Palms formation (quartz-slate).	*Animikie Series:* Upper slate formation. Gunflint formation (iron-bearing, but non-productive).	*Upper Marquette:* Michigamme formation (slate) (locally replaced by Clarksburg volcanic formation). Contains productive iron-ores near base. Ishpeming formation, consisting of two members: Bijiki schist (in western part of district) and the Goodrich quartzite containing productive detrital ores at its base.
	MIDDLE HURONIAN. (Iron-bearing.)				*Middle Marquette:* Negaunee formation (iron-bearing and productive). Siamo slate. Ajibik quartzite.
	LOWER HURONIAN.	Granite intrusive in lower formations. Slate-gray-wacke-conglomerate formation.	Bad River limestone formation (cherty limestone). Quartzite.	Intrusives. Knife slate. Agawa formation (iron-bearing, but non-productive). Ogishke conglomerate.	*Lower Marquette:* Wewe slate. Kona dolomite. Mesnard quartzite.
ARCHEAN OR BASEMENT COMPLEX.	LAURENTIAN. (Intrusive into Keewatin.)	Granites and porphyries.	Granite and granitoid gneiss.	Intrusive granites, porphyries and greenstones.	Granite, syenite, peridotite. Palmer gneiss.
	KEEWATIN. (Iron-bearing.)	Greenstones, hornblende-schists and porphyries.	Green schists and fine-grained gneiss.	Soudan formation (iron-bearing and productive). Ely greenstone, an ellipsoidally-parted basic igneous and largely volcanic rock.	Kitchi schist and Mona schist, the latter banded, and in a few places containing narrow bands of non-productive iron-bearing formation.

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Series of the Lake Superior Region.

Crystal Falls District.	Menominee District.	Keweenaw Point.	Michipicoten.	Other Parts of Ontario North of Lake Superior.
		Middle and Upper Keweenawan. Interbedded lavas and sediments grading up into sandstones.		Interbedded sediments and traps along Lake Superior shore.
Michigamme formation, containing a productive iron-bearing horizon not separated in mapping for much of the district.	*Upper Menominee:* Hanbury formation, mainly slate, bearing in lower portions calcareous slates, etc., containing siderite and iron oxide. Vulcan formation, consisting of three members: Curry member (iron-bearing), Brier slate and Traders member (iron-bearing).			Animikie series of northwest shore comprising iron-formation overlain by slate.
Hemlock formation (basic volcanic). Negaunee formation (iron-bearing).	Negaunee formation doubtfully present.			Undivided pre-Animikie sediments, mainly graywackes, slates and conglomerates, much metamorphosed.
Randville dolomite. Sturgeon quartzite.	*Lower Menominee:* Randville dolomite. Sturgeon quartzite.		Basic eruptives. Acid eruptives. Doré conglomerate.	
Granite.	Granites and gneisses.		Granites and gneisses.	Granites and gneisses.
	Quinnesec schist. (?)		Eleanor slates. Helen formation (iron-bearing and productive). Wawa tuffs. Gros Cap greenstones.	Greenstone and Iron formation, similar to that of Vermilion district of Minnesota.

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International Committee.) The acid rocks, including granites, gneisses and acid schists, are called "Laurentian."

One of the conspicuous features of the greenstones and green schists of the Keewatin is the presence of ellipsoidal, spherulitic and pyroclastic phases, indicating their essential surface-volcanic origin. The basement upon which they were deposited has not been found, unless possibly it be certain of the granites which have been called Laurentian. However, the granites for the most part are certainly later intrusives. It is an interesting fact, then, that the oldest rock in the Lake Superior region (and perhaps the oldest pre-Cambrian rock with definitely determined age in the country) is of surface-volcanic origin, with the basement upon which it was deposited unknown.

The nature and distribution of the Archean rocks in the different parts of the region are discussed in subsequent paragraphs.

Michigan and Wisconsin.—The predominating rocks of the Basement Complex in Michigan and Wisconsin are granites and gneisses which appear both north and south of the Marquette district; within the Crystal Falls district, and to the east of it; to the north and south of the Menominee district; and south of the Penokee-Gogebic district. These granites and gneisses show a variety of characters and are certainly not all of the same age, although so far as known they antedate the Algonkian rocks. One of the acid units on the south side of the Marquette district has been discriminated under the name "Palmer gneiss." The character of this rock is such as to suggest that it may represent, in part, a much-altered acid-sediment. It is certain that rocks mapped as Palmer gneiss are sedimentary, but such sedimentary rocks may really belong to the Algonkian, and may have been included in the Palmer gneiss because of errors in mapping. How far into Wisconsin these Basement-Complex granites and gneisses extend is not known. In north central Wisconsin granites of apparently similar characters have been found to be later than the Basement Complex, and it is not known where, in the great granitic area extending from central Wisconsin to the State boundary, to draw the line between the granites of the Basement Complex and later granites.

Taking up the Keewatin rocks of Michigan and Wisconsin,

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we find on the north side of the Marquette district a considerable area of peridotite and areas of green schist, including chlorite and mica-schist, resulting from the alteration of basalts, some of them presenting evidence of surface-volcanic origin and perhaps of deposition under water. Where finely schistose the rocks have been called the "Mona schists," and where giving evidence of derivation from surface-volcanics they have been called "Kitchi schists." Associated with the green schists of the Marquette district are several small areas of jasper and carbonate rocks, formerly supposed to be of the nature of ferruginous vein-material, but now believed to represent a sedimentary iron-formation infolded or interbedded with the greenstones and belonging with the Basement Complex.

Green schists of similar character occurring both north and south of the Menominee district have been known as the "Quinnesec schists." To the west of the Menominee district along Pine river, apparently similar schists are associated with dark-green sedimentary slates which it has not been possible to separate from them. The same may be said of the slates associated with the green schists along the Brulé river on the north side of the greenstone schist area. These Brulé river slates are adjacent, however, to a great Upper Huronian slate-area and presumably are themselves of this age. The Quinnesec schists have been regarded as Keewatin because of their lithological similarity with Keewatin green schists in the Marquette district, and because of their occurrence on the outer edge of the Menominee-Huronian area, which is thought to have a synclinal structure. But nowhere are direct relations to be observed, and accordingly the relations of the Quinnesec schists to the slates of probable Upper Huronian age on the Brulé river suggest that the green schists themselves may be of Upper Huronian rather than Keewatin age.

South of the Penokee-Gogebic district are two areas of green schist, including mica-schist, chlorite-schists, and hornblende-schists, which have been known respectively as the eastern and western green schist areas. Between the Gogebic and Crystal Falls districts, surrounded by Huronian sediments, are several areas of greenstone and green schist, which probably belong with the Keewatin, but which because of their isolation do not allow of their structural relations and age being determined.

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Central and Eastern Minnesota.—In Minnesota, along the Minnesota, Mississippi and Snake rivers, are areas of granite, gneiss, hornblende-schist, mica-schist, diabase, gabbro, etc., which have been in the past referred to the Basement Complex. Hall, however, concludes that certain of the schists have resulted from the intrusion of granite into the Huronian sediments, as typically developed at Carlton and Cloquet; and thus that most of the acid igneous rocks are of Algonkian age. Certain of the massive diabases running eastward from the Mississippi river are older than the granites, and lithologically resemble the Keewatin greenstones in the Mesabi and Vermilion districts. These are accordingly mapped as Keewatin, while the granites are mapped as Algonkian.

Mesabi District, Minn.—The Archean is represented by several comparatively small areas of greenstone and green schist, comprising ellipsoidal basalt, hornblende-schist, chlorite-schist and mica-schist, intrusive into which are granite porphyries in small quantity.

Vermilion District, Minn.—In this district, the Archean is mainly represented by great areas of ellipsoidal basalt, in some cases showing surface-textures, occurring in great oval-shaped areas surrounded by sediments. Closely associated with these greenstones, in relatively small lenses and bands, are the rocks of the productive Soudan iron-formation, including jaspers, iron-ore and slate. The relation of the iron-formation and greenstone have been much studied, but with only partly positive results. The iron-formation is almost certainly sedimentary because of the association with slates and because of the similarity to other known sedimentary iron-formations of the Lake Superior region; yet, with the exception of a trivial amount of slate at one or two places, the jaspers rest with sharp contact against the greenstone without intervening fragmental material. Mining-operations, together with the surface-distribution of the iron-formation, indicate that the iron-formation, in many cases at least, rests on top of the greenstone and has been infolded with it; but for many of the iron-formation bands, there is little opportunity to determine the relations; and it has been thought probable that some iron-formation bands may be interbedded with greenstone-flows. In this connection it is interesting to note that at Schreiber bay, on the

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north shore of Lake Superior, well-bedded Animikie iron-formation rests on granite without intervening fragmental sediments, although a short distance away this material is found. Intrusive into the Vermilion greenstones and iron-formation are granite-porphyries and massive granites. The latter occur for the most part in the great area lying north and east of the Vermilion range. Immediately adjacent to the range they are known as the Vermilion, Trout, Burntside, Basswood, and Saganaga granites. Along the contact of the greenstone there are considerable belts of hornblende-schist which are taken to represent the contact effects of the intrusion of the granite. This granite presumably extends northward into Canada and connects with the granites mapped by the Canadian geologists; but again, information is not available which will allow the drawing of boundaries between granite belonging to the Basement Complex and granite of later age, some of which (and perhaps a considerable amount) appears in the Canadian districts to the north.

Michipicoten District.—The Archean of this district presents close similarities to the Archean or other parts of the Lake Superior region, especially in the Vermilion and Marquette districts. It consists of greenstones with ellipsoidal and other surface-structure; of tuffs, both basic and acid; of an iron-formation; and of slate,—all intruded by granites and gneisses. The iron-formation is believed to be the counterpart of that seen in the greenstone in the streets of Marquette, in the Vermilion district, and elsewhere in the region.

Other Points on the North Shore of Lake Superior.—Greenstones, with associated iron-formation, similar in many respects to those of the Michipicoten and Vermilion districts, are known in many areas in Ontario. They are typically developed in the Mattawan, Atikokan, and Steep Rock Lake iron-ranges, northwest of Lake Superior.

Algonkian.

The fullest succession of the Algonkian system comprises four unconformable sedimentary series, all of them being associated with igneous rocks. The three lower series are collectively referred to as the Huronian, and the upper series as the Keweenawan. All are unconformably above the Base-

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ment Complex and below the Paleozoic for this district, but in no single district are all of them represented. While there are similarities between the Huronian series of the different districts which are believed to warrant their correlation, the differences are such as to make it desirable to discuss the Huronian succession of the individual districts separately, in order to avoid the impression of definiteness or finality of correlation which would inevitably tend to obscure or perhaps distort the facts. The Keweenawan series is essentially a unit for the region and it is so described on a subsequent page.

Marquette District, Mich.—In the Marquette district, all three of the Huronian series are present. They are locally called the Lower Marquette, the Middle Marquette and the Upper Marquette series. Heretofore the Huronian has been mapped as comprising two divisions,[4] the Upper Marquette and the Lower Marquette series; but the discovery of a great unconformity within the former Lower Marquette by Prof. A. E. Seaman proves the presence of three series. The Lower Marquette series as now defined consists of three formations, the Mesnard quartzite, the Kona dolomite, and the Wewe slate, varying from 1,200 to 3,000 ft. in thickness. The Middle Marquette series includes from the base up the Ajibik quartzite, the Siamo slate, and the Negaunee iron-bearing formation, varying from 1,900 to 3,000 ft. in thickness. The Negaunee iron-bearing formation carries the important iron-ore deposits of the district. The Upper Marquette series includes the Goodrich quartzite, the Bijiki schist, and the Michigamme slates. The Bijiki schist, appearing principally in the western part of the district, is taken to be partly equivalent in age to the Goodrich quartzite, which occurs especially in the central portion of the district; but where the two come together, the Bijiki schist is above the Goodrich quartzite. Both are conformably overlain by the Michigamme slate. There are iron-ore horizons at the base of the Goodrich quartzite, at the base of the Bijiki schist, and at the lower horizon of the Michigamme slate, those at the first horizon mentioned being largely detrital and derived from the underlying Middle Marquette series. Both the Lower and Middle Marquette series

[4] *Monograph XXVIII., U. S. Geological Survey.*

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are intruded by basic igneous rocks in the form of dikes and bosses. The Upper Marquette also is interbedded in the central part of the district with a series of fragmental basic volcanics known as the Clarksburg formation. The Lower Marquette series is the most folded and metamorphosed, the Upper Marquette series the least; the Lower Marquette series is the thinnest of the series, the Upper Marquette the thickest; the Lower Marquette occupies the least area, the Upper Marquette the greatest. The unconformities between the Middle and Lower Marquette and between the Upper and Middle Marquette series are in each case evidenced by basal conglomerates and by truncation of the underlying series. It cannot be said at present which of the two unconformities is the more important.

The structure of the sedimentary series is that of a fan-shaped or abnormal synclinorium with east-west axis and westward pitch. Faulting plays a subordinate part, but still is important. The nature of the faulting is illustrated in the Teal Lake range of the Marquette district and the Teal Lake type of faulting has been found to explain the shape and distribution of certain of the ore-deposits.

The Crystal Falls District, Mich.—In the Crystal Falls district the Huronian rocks have been divided in the U. S. Geological Survey map and report[5] into the Upper and Lower Huronian, corresponding to the old division of Upper and Lower Marquette series for the Marquette district. In the Crystal Falls district the Lower series has not been found to be divided by unconformity, but it includes formations similar to those of the two lower divisions, now known, of the Huronian of the Marquette district, and therefore the three-fold classification will be here adopted. The Lower Huronian consists, from the base upward, of the Sturgeon quartzite and the Randville dolomite, varying from 600 to 1,600 ft. in thickness. The Middle Huronian consists of the Negaunee iron-formation. Cutting and resting on both series is the Hemlock formation, consisting largely of basic volcanics. The Upper Huronian comprises the Michigamme slate-formation, containing an iron-bearing horizon not separated in mapping for much of the district.

[5] *Monograph XXXVI., U. S. Geological Survey.*

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There is practically no evidence of unconformity between the Hemlock volcanics and the Upper Huronian slates, the existence of the break being largely inferred in the Survey report from the difference in lithological character and the apparent close relations of the Hemlock volcanics with the Lower Huronian sediments. There is also no decisive evidence that the iron-formation of the Crystal Falls and Amasa areas, mapped and described as Upper Huronian, may not really be Middle Huronian (Negaunee). It seems to be nearly connected through magnetic lines with iron-formation mapped as Negaunee in the eastern part of the district.

The series have been folded into complex folds with trend varying from the north-south to the east-west. The closer folds in the northeastern part of the district are nearly north-south; in the central part northwest-southeast; in the eastern and southeastern parts nearly east-west. All of these folds have steep pitches.

Menominee District, Mich.—The classification has been in the past Lower Menominee and Upper Menominee, separated by unconformity,[6] but as the formations are similar to those of the Marquette district a three-fold classification is here adopted. The Lower Menominee series comprises the Sturgeon quartzite and Randville dolomite formations, in all respects similar to formations of this name in the Crystal Falls district, and similar, respectively, to the Mesnard quartzite and the Kona dolomite of the Lower Marquette series of the Marquette district. The Middle Menominee series has not been found, although iron-formation pebbles, supposed to represent it, are found in the conglomerate at the base of the next overlying series. The Upper Menominee series comprises the Vulcan formation, consisting in ascending order of three members, (*a*) Traders iron-bearing formation, (*b*) Brier slate, (*c*) Curry member (iron-bearing), and above this the Hanbury slate, bearing in lower portions calcareous slates, siderite and iron oxide. The thickness of the series varies from 2,650 to 3,650 feet. As indicated on a previous page, the basic volcanics (Quinnesec schists) of this district (with associated slates on the Pine and Brulé

[6] See *Monograph XLVI., U. S. Geological Survey.*

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rivers), mapped and described as Archean by the U. S. Geological Survey,[7] may really be Huronian.

In the Menominee district, the structure has been mapped as that of a northwest-southeast trending synclinorium with a minor anticlinal roll bringing up the older formations in the central portion of the district. However, attention is directed to the fact that the Lower Huronian, which ought to appear on the south side of the trough, is here missing, and the synclinal structure is inferred rather than actually observed. The absence of the Lower Huronian here has been explained by the overlapping of the Upper Huronian slates. If the Quinnesec schists to the south should be shown to be Huronian intrusives, rather than Archean basement rocks, then we might look for the southern edge of the Menominee synclinorium almost anywhere to the south, and the suggestion is made that the quartzite to the south of the Quinnesec schists in Wisconsin may really constitute the southern margin of this trough.

Penokee-Gogebic District, Mich. and Wis.—In the Penokee-Gogebic district, representatives of the Lower and Upper Huronian series with intervening unconformity are found. The Middle Huronian seems to be lacking. The Lower Huronian series comprises a lower quartzite overlain by a cherty limestone-formation (Bad River limestone) similar respectively to the Mesnard quartzite and the Kona dolomite of the Lower Marquette series. The Upper Huronian series (the Penokee-Gogebic series) comprises, from the base up, the Palms quartz-slate formation, the Ironwood iron-bearing formation and the Tyler slate formation, doubtfully aggregating 13,300 ft. in thickness. Much the largest portion of the district is occupied by the Upper Huronian rocks, principal in mass among which being the slate. Intrusive into the Upper Huronian are diabase dikes, probably of Keweenawan age. The principal series of the district, the Upper Huronian, strikes in general east-northeast and dips 65° to the north.

Area East of Penokee-Gogebic District.—In the broad triangular area between the east end of the Penokee-Gogebic district and the Marquette, Crystal Falls and Menominee districts, is a great slate formation constituting the upper division of the Upper

[7] *Monograph XLIII., U. S. Geological Survey.*

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Huronian series. In isolated areas within this slate appear iron-formation, productive at Iron River and Florence. There appear also basic volcanics and older Huronian sediments, the correlation of which is doubtful.

Area South of Menominee District.—Extending southwest from the Menominee district, into north central Wisconsin, is an area of quartzites with indefinite limits, which include considerable masses of igneous material of unknown age.

North Central Wisconsin.—Here granites occupies about 75 per cent. of the area of crystalline formations, but associated with it are greenstones, rhyolite, schist, slate, quartzite and conglomerate, believed by Weidman to represent at least two, and perhaps three, of the Huronian series. The results of his work will be published in a bulletin of the Wisconsin Geological and Natural History Survey.

Chippewa River and in Barron County, Wis.—In this region are quartzites and conglomerates, containing fragments of chert and jasper, which again are similar to those of the Huronian of Michigan, but which cannot be assigned to any one division of the Huronian. Their lithological similarities are with the Upper Huronian.

Central Minnesota.—In the vicinity of Carlton, Cloquet and Little Falls, and extending to the southwest, is a considerable area of well-banded graywacke and slate similar in all respects to the undivided Lower Huronian of the Mesabi and Vermilion districts to the north. To the south and southwest are certain hornblende- and mica-schists which Hall[8] believes to be the result of metamorphism of the graywacke and slate by intrusive granite which appears in that direction, but some of them are doubtless metamorphosed igneous rocks.

Near Dam lake, south of Kimberly, is a quartzite exposure which has been correlated with the Upper Huronian of the Mesabi district.

Along the Minnesota river are quartzites similar to the Huronian quartzites of Michigan. Their lithological similarities are with Lower Huronian quartzites, but because of their isolation they are so assigned only tentatively.

Mesabi District, Minn.—In this district, a two-fold classification of the Huronian series is possible. The upper division

[8] Hall, C. W., *Bulletin U. S. Geological Survey*, No. 157.

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corresponds with the Upper Huronian of Michigan and Wisconsin, while the lower division might be equated with either or both of the Lower and Middle Huronian of Michigan so far as present evidence goes. It will be here referred to as the Lower Huronian, following the Survey report.[9] The unconformity between the Upper and the Lower series is, as usual, a great one. The Lower Huronian consists of conglomerates, graywackes and slates, with bedding and schistosity standing vertically and striking east-northeast. It occurs in but small areas, and is intruded by the Giant's Range granite, which constitutes the main topographic feature of the Mesabi range. The Upper Huronian (Mesabi or Animikie series) comprises from the base up, the Pokegama quartzite formation (200 ft. thick), the Biwabik iron-bearing formation (1,000 ft. thick) and the Virginia slate-formation (great, but unknown, thickness). This series rests on the south slope of the Giant's range, strikes east-northeast, dips for the most part southerly and southeasterly at angles varying from 5° to 20°, and is gently cross-folded, the axes of the cross-folds pitching in the direction of the general dip noted. Intrusive into this series at the east end of the district are Keweenawan granite and basic igneous rocks, the latter in the form of sills, laccolites and dikes. Here the series is highly metamorphosed.

Vermilion District, Minn.—Here again only a two-fold classification of the Huronian is possible, these divisions corresponding very closely in lithology and structure with the same divisions of the Mesabi district, although varying greatly in relative areas. The Lower Huronian sedimentary series is composed of conglomerate, slate and graywacke, occupying large areas in this district. Usually the series appears in belts between, and surrounding, the cores of greenstone belonging to the Basement Complex. The schistosity and bedding stands nearly vertical and strikes east-northeast. This series is intruded by granite masses such as the Giant's Range, Cacaquabic Lake, White Iron Lake and Snowbank Lake granites. The Upper Huronian series occurs only in the east end of the district, where the eastward continuation of the Upper Huronian of the Mesabi district laps diagonally and unconformably across the east end of

[9] *Monograph XLIII., U. S. Geological Survey.*

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the Vermilion district. The Upper Huronian here is similar in all respects to that of the metamorphosed Upper Huronian of the east end of the Mesabi district proper.

International Boundary and North Shore of Lake Superior.—Extending from the east end of the Vermilion district in the vicinity of Gunflint lake, eastward to Pigeon Point, and northeastward to the head of Thunder Bay on Lake Superior, and thence in discontinuous areas past Nepigon Bay, is a great area of Upper Huronian or Animikie rocks, similar in all respects to the Upper Huronian rocks of the Mesabi district and of the south shore, except that the iron-bearing and quartzite-members are not so well developed and the series is more largely intruded, parallel to the bedding, by Keweenawan sills of basic igneous rock, known as the Logan sills. Lower Huronian sediments have been recently found to appear in a narrow belt north of the Animikie rocks to the east of Thunder Bay.

Michipicoten District.—Rocks of Lower Huronian age consisting of conglomerate, graywacke and slate, associated with basic and acid eruptives, similar in all respects to the Lower Huronian series of the Vermilion district are here present.

KEWEENAWAN.[10]

The uppermost of the four members of the Algonkian series of this region, the Keweenawan, is a series of coarse, fragmental sediments, sandstone and conglomerate, with small amounts of limestone, interbedded and cut with basic and acid igneous rocks, including gabbro, diabase, porphyrite, amygdaloid, felsite, quartz-porphyry, etc. For the district as a whole, the igneous rocks are more abundant in the lower portions than in the upper portions. Accordingly a three-fold division has been made, from the base up, into gabbros, interstratified lavas and clastic rocks, and clastics. On the north shore, however, eastward from Port Arthur, sediments in considerable volume constitute the lowest part of the series. Many of the basic and acid rocks are surface flows. The diabase-flows frequently pass in their upper parts into porphyrites and amygda-

[10] This description follows somewhat closely, except for certain new developments, the one given by Van Hise in his paper, The Principles of North American Pre-Cambrian Geology, *16th Annual Report U. S. Geological Survey*, Pt. I., pp. 571–874.

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loids. Many flows are porphyritic or amygdaloidal throughout. The beds of quartz-porphyry and felsite are abundant in certain districts, but usually have no great lateral extent. While a single basic or acid flow may be traced but a little way, frequently a set of flows of the same general character may have a great extent and thickness. But even the sets of flows cannot be regarded as general formations for the whole of the Lake Superior basin. No tuffs have as yet been found.

The Keweenawan rocks occur for the most part around the west end of Lake Superior with a general synclinal structure, the rocks of the series on the south side of the series dipping somewhat steeply to the north and those on the north side dipping less steeply toward the south.

On the south side of the lake, the principal development of the series is on Keweenaw point, whence the series gets its name. Here are located the important copper-bearing mines of the Lake Superior region. The copper occurs as metallic copper in amygdules of the flows, and replacing parts of both the igneous and clastic rocks, especially the conglomerates. The upper part of the series here is almost clear sandstone.

The Keweenaw point series rests on the southeast against Upper Cambrian sandstone, there being along the contact a marked fault-plain which has obscured the relations of the two. While there has been difference of opinion on the interpretation of the facts, it is the prevailing view that a pre-Cambrian fault-scarp with downthrow toward the east here existed, against which the Cambrian sediments were deposited, and that subsequent to their deposition further faulting in the same direction broke and deformed the Cambrian rocks near the contact.

To the southwest of Keweenaw point, the Keweenawan series comes in contact with the Huronian of the Penokee-Gogebic district,—in the central part of the district with the slate and at the east and west end of the district with the iron-formation of the Upper Huronian series, indicating that, prior to the deposition of the Keweenawan series in this district, there had been folding of the Upper Huronian series in such a way as to expose the east and west ends to erosion; that here the slate had been eroded away, and that subsequently the Keweenawan rocks were deposited against the truncated edges of the series. The lowest rocks are, for the most part, igneous flows dipping

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in much the same direction as the Upper Huronian, but in or at certain places in the lower horizons conglomerates are taken to represent the base of the series.

Still farther to the west and southwest, the Keweenawan series extends through Douglas county, Wisconsin, into Minnesota. The characters of the series are much the same as on Keweenaw point. Copper-deposits are present, but of disputed value. On the northwest side of the tongue of the Keweenawan projecting southwest into Minnesota, there is again a contact with Cambrian sediments along which there has been marked faulting tending to obscure the relations. The Cambrian rocks for the most part are turned up and broken along the contact.

On the northwest shore of Lake Superior, the Keweenawan series consists almost wholly of the lower igneous portion. The most important member is a great gabbro mass extending from Duluth to the international boundary, in a great crescent with concavity toward the lake, having a width for much of the distance of from 20 to 25 miles. The rock has everywhere a typical granitic structure, and includes, lithologically, very magnetitic gabbro, olivine-gabbro, normal gabbro, and masses of basic feldspar rocks or anorthosite. The relations of these varieties to one another have not always been determined. Much of the gabbro has a laminated arrangement of the mineral particles, and corresponding with this arrangement the rock has a bedded or sheeted appearance. Overlying this gabbro mass are flows of both basic and acid eruptive rocks, ending diagonally against the gabbro, and, near the international boundary, associated with these flows are conglomerate (Puckwunge conglomerate) masses, probably interbedded with the Keweenawan flows. The flows also have crescentric distribution to the southeast of the gabbro. On the northwest the Keweenawan gabbro overlaps the east end of the Mesabi district, lapping diagonally across all the members of the Upper Huronian series onto the Lower Huronian granites. Near Gunflint lake and eastward along the international boundary, and northeast beyond Port Arthur, basic sills of the Keweenawan have penetrated the Upper Huronian series parallel to the bedding. The metamorphic effects of the Keweenawan igneous rocks on the Upper Huronian series have been most profound, producing cordierite-

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hornstones from the slates, amphibole-magnetite rocks from the iron-formation, and very coarsely crystalline quartzites from the quartzite. Moreover, by minute injection and transfer of material at the contacts other peculiar metamorphic rocks have been developed at Pigeon point, Gunflint lake, and elsewhere. Near Loon lake, northeast of Thunder bay, the basal horizon of the Keweenawan is a course conglomerate resting in turn on Upper Huronian and Lower Huronian formations and carrying fragments of them.

The relations of the Keweenawan to the underlying rocks are somewhat complex because of the combined sedimentary and igneous nature of the series. On the south shore, as already indicated, the relations are for the most part confused by faults. In the Penokee-Gogebic district, conglomerates of doubtful horizon may represent the base of the series. At various places on the north shore of Lake Superior, as on the northeast side of Thunder bay, the lowest member of the Keweenawan series is a conglomerate containing fragments of the underlying rocks, which here consist both of granite and greenstone and Upper and Lower Huronian sediments. The conglomerate is associated with sandstone and limestone. Together the sediments are in considerable mass. To the west of Port Arthur, they diminish in amount, and in the Mesabi, Gunflint Lake, and international boundary areas, the lowest rock of the Keweenawan is the Duluth gabbro. It was formerly supposed that the Duluth gabbro represented a great basic flow, which was poured out at the beginning at Keweenawan time, covering the truncated edges of the Animikie and underlying series. It is now believed that it represents a great intrusion which occurred in late Keweenawan time between the interbedded sedimentary and igneous flows of the Keweenawan series, and the underlying pre-Keweenawan rocks. The following characteristics favor this view,—coarse texture of the mass; lack of surface volcanic textures; apparent connection with the Logan sills which are intruded into the Upper Huronian sediments; minute injection of gabbro material into the iron-formation in the vicinity of Akeley and Gunflint lakes; profound metamorphism which it has caused in the adjacent rocks; diagonal ending of overlying Keweenawan flows against the gabbro, which on the theory of extrusive

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nature of the gabbro would require erosion, or tilting between the gabbro and the flows; and, finally, the fact that, for at least a part of the district, the Keweenawan period was inaugurated by the deposition of sediments rather than by igneous activity.

The Keweenawan is the thickest of the series about Lake Superior. Its average was estimated by Irving on Keweenaw Point to be 35,000 ft., but estimates by subsequent observers are less.[11] This vast quantity of material does not, of necessity, mark a period longer than, or perhaps even one as long as, the Lower Huronian or Upper Huronian, since the greater part of it is of igneous origin, and doubtless accumulated rapidly. The lava-flows in their extent and thickness are to be compared with the great volcanic plateaus of the far West rather than with local volcanoes, such as Vesuvius, or the local volcanoes of the Upper Huronian and Lower Huronian. No volcanic fragmental material associated with the lavas has as yet been discovered.

The fact that erosion was contemporaneous with eruption for much of Keweenawan time is to be noted. Certainly, when the period was well inaugurated, much of the Lake Superior basin was frequently, if not normally, below the sea or near tide-water. Many of the eruptions may have been sub-aqueous. Here and there volcanic masses of such magnitude were built up as to rise above the water, and upon such areas the sea at the base, and the air and rain above, immediately began their course of destruction. The acid and more viscous lavas may have formed the more prominent elevations, and thus the attack was here more vigorous. This may partly explain the predominance of the acid pebbles in the conglomerates. Prof. Seaman has called attention to the manner in which similar acid slags puff up and suggests, that because of this, the acid lavas may have yielded more readily to the waves.

This great volcanic period was doubtless one of unstable equilibrium, the lithosphere falling here and rising there. The final resultant of all the movements was the production of the Lake Superior syncline. This synclinal movement affects not only the Keweenawan rocks, but also the lower series, and in

[11] Burwash estimates thickness on Michipicoten Island at 36,000 ft. in a recent paper, The Geology of Michipicoten Island, by E. M. Burwash, University of Toronto Studies, Geological Series No. 3, Toronto, 1905.

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areas in which the unconformity between the Upper Huronian and the Keweenawan is not great, there is such a likeness in strike and dip of the two series as to suggest at first that the two are conformable. It is only as the contacts between them are followed for some distance, and the Keweenawan is seen to be now in contact with one member of the Upper Huronian and now with another, that an unconformity between the two is perceived.

What proportion of the Keweenawan has accumulated before the Lake Superior syncline began to form, it is impossible to say. Possibly the larger foci from which the great extrusions of lava occurred were in the area now covered by water. Probably sinking went on contemporaneous with volcanic action, as is commonly a result of the upbuilding of mountainous masses of volcanic material. This suggestion, if true, would partly explain the apparent absence of volcanic fragmental material, which naturally would accumulate near the foci. Nowhere are the Keweenawan rocks so closely folded as to give them a secondary structure, or a much altered character.

The diabases, gabbros and diorites, so extensively intruded within the Lower Huronian and Upper Huronian rocks, in part at least, are, in all probability, of Keweenawan age; but this statement is difficult to prove.

General Notes on Structure and Distribution of the Lake Superior Series.

For the region, as a whole, it is difficult to make a brief statement adequately covering the complex structure and distribution of the pre-Cambrian series; but consideration of a few general features may aid in fixing the reader's ideas of the general geology of the region. By mapping under one symbol all greenstones and under another symbol all granites, whether of Archean or Algonkian age, the granite- and greenstone-areas may be easily remembered in their general distribution (see Fig. 1). By so doing we may avoid also the necessity of drawing boundary lines between granite and greenstone of different ages in the parts of the Lake Superior region in which a separation has not been satisfactorily made. For stratigraphic purposes, the granite- and greenstone-areas may be practically eliminated from our consideration. There then

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remain principally the pre-Cambrian sedimentary series with interbedded volcanics which are of primary importance in comprehending the general stratigraphy of the region. These are grouped about the west half of Lake Superior. The west end of Lake Superior consists of an eastward-pitching synclinorium of Keweenawan rocks with gentle southerly dips on its north limb and fairly steep northerly dips on its south limb, the Keweenawan series there fringing the west half of the lake. The next underlying series, the Upper Huronian, to a less extent takes part in this synclinal structure and has a distribution consonant with such structure, that is, bordering the outer edge of the Keweenawan areas. In detail, the structure of the Upper Huronian is far more complex than that of the Keweenawan series; and in Michigan, in the triangular area extending eastward from the Penokee-Gogebic district and fingering out in the Marquette, Crystal Falls and Menominee districts, the Upper Huronian appears in a series of rolls, which apparently have no counterparts in the Keweenawan series. The Lower and Middle Huronian sediments occupy but limited areas as compared with the other Algonkian rocks. In large part they were folded before Upper Huronian or Keweenawan time, and their strikes and dips give no evidence that they form a part of the Lake Superior synclinal, but they appear near the complex outer border of the Lake Superior synclinal as constituted by the Keweenawan and Upper Huronian rocks, or, in other words, with a major distribution required by the general synclinal structure of the lake. In the Marquette, Crystal Falls and Menominee districts, the distribution is more complex. The Archean rocks again give no evidence in their strikes and dips of participation in the Lake Superior synclinal structure, but they also border this synclinal as constituted by the Upper Huronian and Keweenawan rocks.

It has been possible to explain the major structure and distribution of the Algonkian series on the basis of folding. But recent study has tended to show that faulting has played a greater part than had been supposed, although still a subordinate part. A system of definitely recognizable strike and dip-faults has been worked out in the Marquette district by Prof. Seaman, and the effect of these faults on the distribution and structure of the ore-deposits has been noted in at least one of

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the mines. Presumably, further work will show more of such features. In the Gogebic district important faults, both across and parallel to the iron-formation, have been found to have an important bearing on the distribution and structure of the ore-deposits. In the Mesabi range, a normal fault, inferred but not observed during the preparation of the Geological Survey report on the Mesabi range, has been exposed by further mining-operations in the Biwabik mine. Strike faults of considerable magnitude also occur in the Animikie iron-range east of Thunder bay.

As a result of the closer folding of the lower series in the region as a whole, as compared with the higher ones, and as a result also of the intrusion into the lower series of deep-seated, highly metamorphosing, igneous masses to a greater extent than into the higher series, the lower series shows, in general, more profound metamorphism than the higher ones, although all of them are altered.

General Notes on the Correlation of the Lake Superior Series.

The Keweenawan series of Lake Superior region is recognized by all as approximately a unit. The same may be said of the series here called the Upper Huronian series (Animikie of the north shore), although it is possible that the iron-formation in the Menominee district now mapped with the Upper Huronian, will have to be separated from it and placed in the Middle Huronian series. Through Michigan, the great connecting formation of the Upper Huronian is the black slate which occurs in each of the districts and in the area to the west. While the similarity of the iron-formation of the Menominee and Crystal Falls district to the Negaunee or Middle Marquette iron-formation has been recognized, and the structure of the region combined with magnetic work, seems partly to indicate their connection, it has not been possible to find any break between the Menominee and Crystal Falls iron-formations and the great overlying slate-member, and because of this, the iron-formation has been mapped with the black slate, which is Upper Huronian, if anything in the district is Upper Huronian.

The rocks, called Lower and Middle Huronian, show much wider diversity of lithology and corresponding greater difficulty

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in correlation than the overlying series. The two series have been precisely separated only in the Marquette district, but in the Crystal Falls, Menominee and Gogebic districts the sedimentary formations beneath the Upper Huronian are so similar to formations of the Lower and Middle Marquette series, that there seems little doubt of their substantial equivalence. On this basis, both the Lower and Middle Huronian are represented in the Crystal Falls district; the Lower Huronian is represented in the Menominee district, and perhaps also the Middle Huronian, if the iron-formation proves here to belong; and the Lower Huronian is represented in the Gogebic district. In the Vermilion, Mesabi, Carlton and Cloquet districts, are sediments recognized by all as belonging to one series below the Animikie or Upper Huronian series, and above the Archean. This series is not similar in lithology to the Lower or Middle series of Michigan and Wisconsin. The term Lower Huronian, which has been applied to it, signifies only that the series is like the Lower Huronian series of Michigan and Wisconsin in being beneath the Upper Huronian. The Minnesota Lower Huronian may represent either or both of the Middle and Lower Huronian series of Michigan. This lower series of sediments in Minnesota has been usually referred to by the Minnesota Survey as the Upper Keewatin series, although not sharply separated in mapping and including members which the U. S. Geological Survey had not included in its Lower Huronian. The conglomerates and slates of the Michipicoten and other districts of Ontario are correlated with the Lower Huronian series of the Vermilion district of Minnesota on the basis of their striking similarity in lithology, structure, and in relations to the Archean.

The discovery by Professor Seaman that the Lower Huronian series of the Marquette district really comprises two great series, which are here called the Middle and the Lower Huronian series, has led to a revision of the correlation of the Lake Superior and Lake Huron series. Prior to this discovery, emphasis had been placed on the dual nature of the series, and this had involved a correlation of the undivided Lower and Middle series of Michigan with the lower series of the Original Huronian district on the north shore of Lake Huron, and a correlation of the Upper Huronian series of the Lake Superior

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region with the upper of the two unconformable series on the north shore of Lake Huron. It now seems probable that the Upper series of the Lake Superior region is lacking in the Lake Huron area, and that the two series there found should be correlated with the Middle and the Lower series of the Marquette district. The importance of this change from the standpoint of correlation is a very great one.

The correlation of the rocks here called Archean or Basement Complex has few doubtful features in the region, although the nomenclature is not generally agreed upon. The basal position of certain granites in Michigan and northern Minnesota has been recognized by all. But in the past the U. S. Geological Survey had included with the basal granites certain of the granites which they now include in the Lower Huronian, in this coming more nearly in accord with Minnesota and Canadian geologists who have held that more of the granites are later intrusives than belong to the Basement Complex.

There are many similarities in the Keewatin greenstones and green schists of the Michigan, Wisconsin, Minnesota, and Ontario districts, perhaps the most striking being the presence of ellipsoidal structures and pyroclastic phases. There is also similarity between them in their association with sedimentary slate and iron-formation material. In previous work, the U. S. Geological Survey has supposed the iron-formation material in the Marquette district to represent ferruginous vein-material and to have no stratigraphical significance; also, in the Vermilion district, it had mapped the iron-formation as Lower Huronian. When iron-formation was found to be unconformably below the base of the Lower Huronian of this district, it was referred to the Archean with the igneous complex previously so referred. The term Archean or Basement Complex had come to be generally regarded as one applying strictly to a basal igneous complex, and the term Algonkian was regarded as including all of the sedimentary rocks above such a complex and below the Cambrian. The reference, therefore, of sedimentary iron-formation to the Archean was met with much opposition. However, the term Archean by definition has never been intended to apply strictly to igneous rocks. It has been held that the Archean complex is mainly igneous, although the possible sedimentary origin of some of it has been recog-

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nized, as in the case of the Kitchi schists and Palmer gneiss of the Marquette district. It has been meant to cover the members of the Basement Complex, not yet stratigraphically separable, beneath the well-recognized sedimentary series mapped Algonkian. As indicated on a previous page, the relations of the Vermilion iron-formation to the associated igneous rocks is not definitely known; and so intricately is the formation intermingled with these igneous rocks that the areas mapped as iron-formation really contain a much larger amount of greenstones than of the iron-formation rocks. Under these circumstances it was thought best in the mapping and description still to refer the iron-formation to this Basement Complex, with which it is so closely associated. These recent developments have meant simply that the Archean rocks have been shown to be in small part of sedimentary origin, a possibility which has never been denied, but developments have not yet shown the advisability of attempting to separate this sedimentary portion and take it out of the Archean. If such were done with the iron-formation, equally strong arguments could be offered for separating certain rocks of doubtful though supposed partly fragmental origin, such as the Palmer gneiss and the Kitchi schist of the Marquette district, a separation which, in the present state of knowledge concerning the Basement Complex rocks, would be considered as premature. By keeping these rocks under one great division separated by a marked unconformity from the clearly recognizable and definable sedimentary formations above, the problem of the correlation of the Archean and Algonkian rocks is simplified. After sufficient definiteness has been reached in the major correlation it may be advisable to attempt further separations from the Basement Complex.

The simplest and clearest conception of the pre-Cambrian succession for the Lake Superior region may perhaps be obtained by thinking of it as primarily divided into six series—Keweenawan, Upper Huronian, Middle Huronian, Lower Huronian, Keewatin and Laurentian—all but the last two separated by unconformities of varying and disputed relative importance, and with age-values not yet decided. These series may then be grouped under the two terms Algonkian and Archean, thereby expressing our opinion that the unconformity

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at the base of the Huronian, in connection with difference in lithological characters above and below this horizon, should be singled out for special emphasis. If we believe, as Lawson does, that the essential break is at the base of the Upper Huronian series, his "Eparchean interval," then we should express our view, as he has done, by grouping all of the rocks beneath this unconformity under one major heading.

Since the manuscript of this paper was first written, the joint committee of the Canadian and United States Geological Surveys, appointed to reach an agreement on disputed points of correlation and nomenclature, has completed its report, and recommends this nomenclature, without expressing opinion as to the desirability of a major grouping of the series under Archean and Algonkian. It is agreed further, in recognition of the difficulties of mapping broad granitic and gneissic areas in Canada, that the term Laurentian may sometimes be applied to areas of such rocks which locally may be known to be pre-Huronian, but which for the most part may be of unknown age.

The Iron-Ores.

Nature, Occurrence, Relations to Adjacent Rocks, Origin.

A general summary of the work of the U. S. Geological Survey on the iron-ores of the Lake Superior region is given by Van Hise.[11] The present summary covers much of the same ground, but includes, also, developments subsequent to the publication of his report.

The iron-ores of the Lake Superior region occur as concentrations in "iron-formations," ranging from a few hundred to 1,000 ft. or more in thickness. These formations in their present form represent the alterations of chemically deposited sediments, for the most part interbedded with normal clastic sediments, such as slate and quartzite.

Because of the definite stratigraphical position of the iron-ores, in contrast to many vein-deposits, the mapping and interpretation of the general geology of the region has been found to be of direct and practical value to the mining-industry, thus explaining the large expenditures by mining interests on geological work.

[11] 21st *Annual Report of the U. S. Geological Survey.*

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In the Vermilion and Michipicoten districts the productive iron-formation is in the Keewatin division of the Archean. In the Mesabi and Gogebic districts the iron-formation is a part of the Upper Huronian series. In the Marquette district two productive iron-formations are present, one in the Middle Huronian and another in the Upper Huronian, the former being the more important. In the Crystal Falls district the iron-formation is in both Upper Huronian and Lower Huronian series, the former being the principal producer. In the Menominee district the formation is of Upper Huronian age.

The iron-formations of the different districts and ages are surprisingly similar in their general characters. Indeed, it was long assumed, erroneously, that because of their similarity they must be of the same age and origin. We may then describe them as a unit.

The iron-formation consists mainly of chert, or quartz, and ferric oxide, segregated in bands or shots or mingled irregularly. Where in bands, with the quartz bands colored red and the rock highly crystalline, it is called jasper. Where less crystalline and either in bands or irregularly intermingled, the rock is known as ferruginous chert. In the Mesabi district the local name "taconite" is applied to the ferruginous chert. Other, subordinate, phases of the iron-formation are (1) ordinary clay slates, showing every possible gradation through ferruginous slate into ferruginous chert; (2) paint-rocks, altered equivalents of the slates; (3) amphibole-magnetite schists; (4) cherty iron carbonate (siderite) and hydrous ferrous silicate (greenalite); (5) the iron-ores themselves.

It may be emphasized that almost the entire bulk of the iron-formation now consists of iron oxide and silica, with carbonates and alumina present in subordinate quantity. The varying combination of part or all of these constituents, mechanically, or chemically, or both, gives all of the rock-types above listed.

A series of cross-sections, Figs. 1, 2, 3 and 4, summarizes better than a description the structural relations of the ores to the adjacent rocks in the different districts and formations.

It has been shown that the ferruginous cherts, jaspers, amphibole-magnetite schists, and iron-ores of the iron-formation result from the alteration either of the cherty iron carbonate or of ferrous silicate (greenalite), or, to a minute extent, from iron

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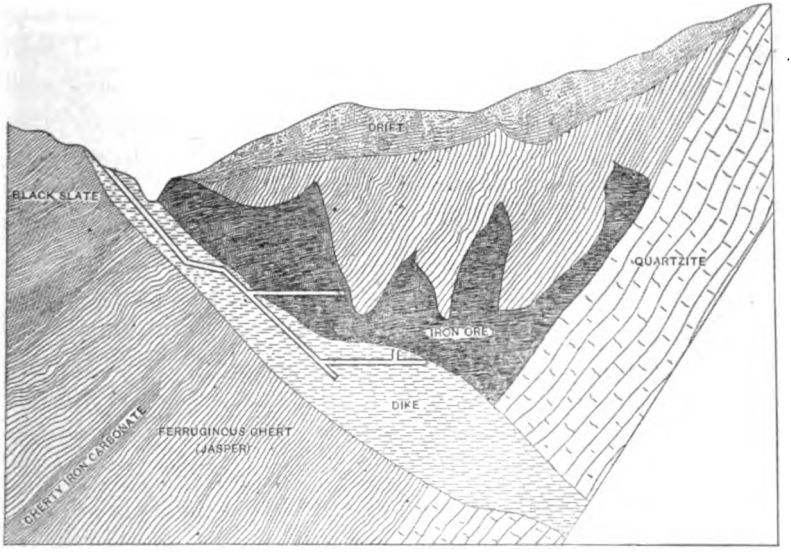
sulphide. The small amounts of iron carbonate or ferrous silicate now found in the formations represent mere remnants left unaltered where so protected by other rocks as not to have been

FIG. 1.

GENERALIZED VERTICAL SECTION IN MARQUETTE DISTRICT, MICHIGAN, SHOWING RELATION OF DIFFERENT CLASSES OF ORE-DEPOSITS TO ASSOCIATED FORMATIONS.

affected by altering agents. The steps of the alteration may be observed, and, in the end-products, the structures and textures of the original rocks are often preserved to a remarkable de-

FIG. 2.



GENERALIZED VERTICAL SECTION THROUGH PENOKEE-GOGEBIC ORE-DEPOSIT AND ADJACENT ROCKS; COLBY MINE, BESSEMER, MICH.

gree. The chemistry of the change is summarized in Table II. Let it be emphasized that the ores and ferruginous cherts or jaspers and the amphibole-schists represent alterations from the

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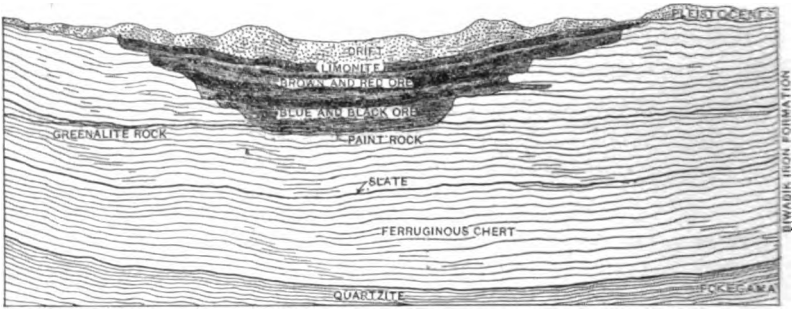
same original types. There is a prevailing notion that the original source of the ores is the jaspers or ferruginous cherts them-

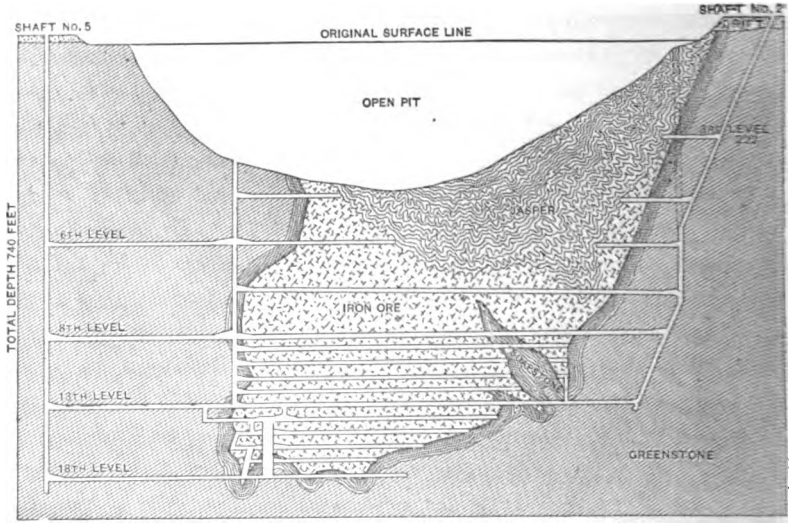
FIG. 3.



GENERALIZED VERTICAL SECTION THROUGH MESABI ORE-DEPOSIT AND ADJACENT ROCKS.

selves, in their present form. While the leaching out of silica from such rocks does yield ore in many cases, it is believed that

FIG. 4.



VERTICAL SECTION THROUGH ORE-DEPOSIT AND ADJACENT ROCKS OF VERMILION IRON RANGE; CHANDLER MINE, ELY, MINN.

ores and jaspers, often of the largest deposits, develop side by side contemporaneously from the alteration of iron carbonates or iron silicates.

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The agent of the alteration is water, coming more or less directly from the surface, carrying oxygen and carbon dioxide. The concentration of the ores has been found to occur where such waters have been converged. Various factors have determined this convergence—fracturing and brecciation of the iron-formation, existence of impervious layers in such attitudes as either to converge waters coming from above or to impound the waters and deflect their course between two layers. The presence of an impervious layer forming a pitching-trough is perhaps the most conspicuous structural feature determining the convergence of waters concentrating the ores. The impervious troughs consist, in the Mesabi district, of slate or paint-rock layers within the formation itself; in the Vermilion district, of greenstone with which the iron-formation is infolded; in the Penokee-Gogebic district, with the intersection of diorite dikes with a foot-wall quartzite; in the Marquette district, of a greenstone intrusive into the iron-formation, or of a slate underlying the iron-formation; in the Menominee district, of dolomite underlying the iron-formation or of slate layers within the formation itself. In all these districts, except the Mesabi, the presence of this impervious basement seems to be clearly the controlling factor in the convergence of waters which have concentrated the ores.

In the Mesabi district, also, impervious troughs may be important, but they probably are subordinate to, or, at least, not more important than, other factors. The iron-formation and its associated rocks lie in beds on the south slope of the Giant's range, and dip off gently to the south at angles averaging from 8° to 10°. In addition to the general southward tilting of the beds, they are gently flexed into folds with axes transverse to the trend of the range. Waters falling on the south slope of the Giant's range, and flowing to the south, enter the eroded edges of the iron-formation and continue their way down along its layers, some of which are pervious and some of which are slaty and comparatively impervious to water. The flow thus tends to become concentrated along the axes of the synclines which pitch gently to the southward. Such synclines are not necessarily surface-troughs. They are evidenced by the attitude of the layers of the iron-formation, and may not be apparent in the unequally-eroded rock-surface or at the surface

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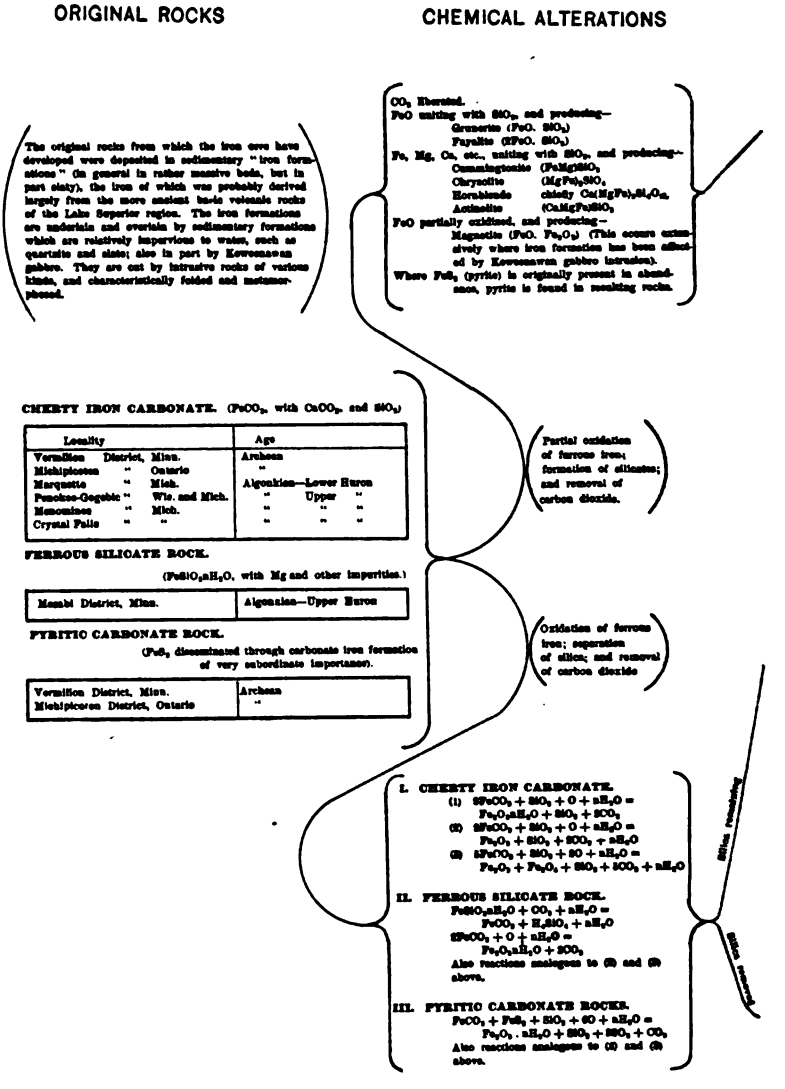
TABLE II.—*Origin and Development of*

(Prepared for the St. Louis Exposition by C. R.


ORIGINAL ROCKS
CHEMICAL ALTERATIONS
The original rocks from which the iron ores have developed were deposited in sedimentary "iron formations" (in general in rather massive beds, but in part slaty), the iron of which was probably derived largely from the more ancient basic volcanic rocks of the Lake Superior region. The iron formations are underlain and overlain by sedimentary formations which are relatively impervious to water, such as quartzite and slate; also in part by Keweenawan gabbro. They are cut by intrusive rocks of various kinds, and characteristically folded and metamorphosed.
CO_2 liberated.
FeO uniting with SiO_2, and producing—
Grunerite (FeO. SiO_2)
Fayalite ($2FeO. SiO_2$)
Fe, Mg, Ca, etc., uniting with SiO_2, and producing—
Cummingtonite ($FeMgSiO_3$
Chrysotile ($Mg_3Si_2O_5$,
Hornblende chiefly $Ca(MgFe)_3Si_4O_{12}$,
Actinolite ($CaMgFeSiO_3$,
FeO partially oxidized, and producing—
Magnetite (FeO. Fe_2O_3) (This occurs extensively where iron formation has been effected by Keweenawan gabbro intrusion).
Where FeS_2 (pyrite) is originally present in abundance, pyrite is found in resulting rocks.
CHERTY IRON CARBONATE. ($FeCO_3$, with $CaCO_3$, and SiO_2)
Locality | Age
Vermilion District, Minn. | Archean
Michipicoten " Ontario | "
Marquette " Mich. | Algonkian—Lower Huron
Penokee-Gogebic " Wis. and Mich. | " Upper "
Menominee " Mich. | " " "
Crystal Falls " " | " " "
Partial oxidation of ferrous iron; formation of silicates; and removal of carbon dioxide.
FERROUS SILICATE ROCK.
($FeSiO_3nH_2O$, with Mg and other impurities.)
Mesabi District, Minn. | Algonkian—Upper Huron
PYRITIC CARBONATE ROCK.
(FeS_2, disseminated through carbonate iron formation of very subordinate importance).
Vermilion District, Minn. | Archean
Michipicoten District, Ontario | "
Oxidation of ferrous iron; separation of silica; and removal of carbon dioxide
I. CHERTY IRON CARBONATE.
(1) $3FeCO_3 + SiO_2 + O + nH_2O =$
$Fe_3O_3nH_2O + SiO_2 + 3CO_2$
(2) $3FeCO_3 + SiO_2 + O + nH_2O =$
$Fe_3O_3 + SiO_2 + 3CO_2 + nH_2O$
(3) $5FeCO_3 + SiO_2 + 3O + nH_2O =$
$Fe_3O_3 + Fe_2O_3 + SiO_2 + 5CO_2 + nH_2O$
II. FERROUS SILICATE ROCK.
$FeSiO_3nH_2O + CO_2 + nH_2O =$
$FeCO_3 + H_2SiO_3 + nH_2O$
$2FeCO_3 + O + nH_2O =$
$Fe_2O_3nH_2O + 2CO_2$
Also reactions analogous to (2) and (3) above.
III. PYRITIC CARBONATE ROCKS.
$FeCO_3 + FeS_2 + SiO_2 + 3O + nH_2O =$
$Fe_2O_3 . nH_2O + SiO_2 + 2SO_2 + CO_2$
Also reactions analogous to (2) and (3) above.
Silica remaining
Silica removed

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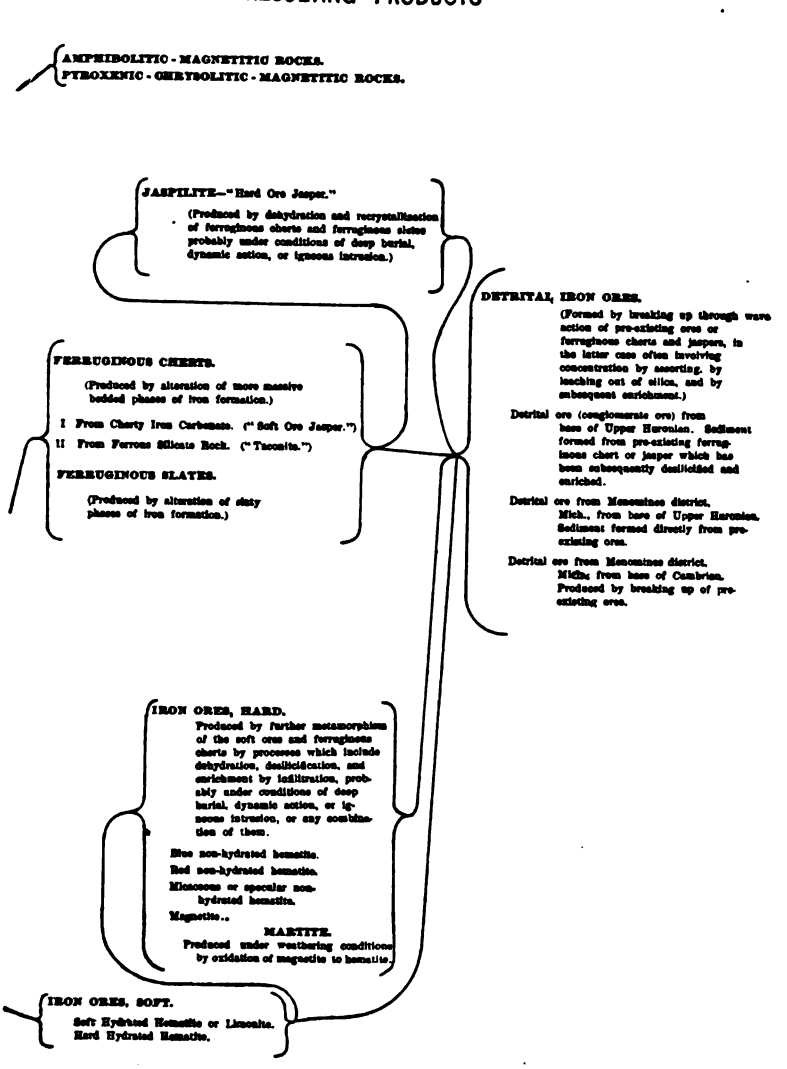
Iron-Ores of the Lake Superior Region.

Van Hise, C. K. Leith and W. N. Smith.)


RESULTING PRODUCTS
AMPHIBOLITIC - MAGNETITIC ROCKS.
PYROXENIC - CHRYSOLITIC - MAGNETITIC ROCKS.
JASPILITE—"Hard Ore Jasper."
(Produced by dehydration and recrystallization of ferruginous cherts and ferruginous slates probably under conditions of deep burial, dynamic action, or igneous intrusion.)
DETRITAL IRON ORES.
(Formed by breaking up through wave action of pre-existing ores or ferruginous cherts and jaspers, in the latter case often involving concentration by assorting, by leaching out of silica, and by subsequent enrichment.)
Detrital ore (conglomerate ore) from base of Upper Huronian. Sediment formed from pre-existing ferruginous chert or jasper which has been subsequently desilicified and enriched.
Detrital ore from Menominee district, Mich., from base of Upper Huronian. Sediment formed directly from pre-existing ores.
Detrital ore from Menominee district, Mich., from base of Cambrian. Produced by breaking up of pre-existing ores.
FERRUGINOUS CHERTS.
(Produced by alteration of more massive bedded phases of iron formation.)
I From Cherty Iron Carbonate. ("Soft Ore Jasper.")
II From Ferrous Silicate Rock. ("Taconite.")
FERRUGINOUS SLATES.
(Produced by alteration of slaty phases of iron formation.)
IRON ORES, HARD.
Produced by further metamorphism of the soft ores and ferruginous cherts by processes which include dehydration, desilicification, and enrichment by infiltration, probably under conditions of deep burial, dynamic action, or igneous intrusion, or any combination of them.
Blue non-hydrated hematite.
Red non-hydrated hematite.
Micaceous or specular non-hydrated hematite.
Magnetite..
MARTITE.
Produced under weathering conditions by oxidation of magnetite to hematite.
IRON ORES, SOFT.
Soft Hydrated Hematite or Limonite.
Hard Hydrated Hematite.

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of the irregular covering of glacial drift. The concentration of the flow along the synclines in the layers of the iron-formation seems so simple and evident that there is a temptation to generalize and say that the underground circulation has probably concentrated the ores along these broad synclines. When the district was first examined by the U. S. Geological Survey this simple explanation was applied. But further study shows that other factors modify the circulation of water and the localization of the ore, and that these secondary factors may be locally dominant.

The most important of these modifying factors is the fracturing of the iron-formation which has furnished numerous trunk-channels for the circulation of underground-waters. The water has been confined to narrow, irregular and most devious trunk-channels formed by the fracturing of the iron-formation, and, while it has probably followed the fracture-openings along synclines to a greater extent than along anticlines, it has not filled the entire syncline formed by the folding of the iron-formation. The result is that the ores have developed along limited and irregular areas within the synclines. They may occupy a considerable part of the syncline, in which case the synclinal structure of the iron-formation may be observed in the layers of wall-rock adjacent to the ores. In other cases, they occupy so small a proportion of the syncline that the layers of the iron-formation in the adjacent wall-rock give no indication of synclinal dips. Not infrequently several more or less independent deposits may have developed in the same general syncline, as, for instance, in the area adjacent to the town of Virginia. To put it briefly, the ores show such position, irregularity, extent, and relations to wall-rocks as to make applicable the expression sometimes heard in the district that the ores have developed through the "rotting" of the iron-formation along fractures, usually, but not always, in broad synclinal areas.

Other factors modifying the general underground-flow of water in the Mesabi iron-formation are the numerous impervious slaty layers within the iron-formation, and the Virginia slate capping the iron-formation of the south; all of which have considerable effect in directing water-circulation. So far as the water is free to flow southward through the iron-formation, the

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impervious layers serve only to limit the flow below. But the continuous south dip of the impervious strata carries the waters down to a point when the ground is saturated and the waters are ponded between impervious layers above and below. That ponding actually occurs is shown by the fact that drill-holes penetrating the slates and entering the iron-formation frequently meet water under pressure, indicating artesian conditions. When ponded, the water seeks the lowest point of escape, which is likely to be found near the north margin of the slate-layers. The movement of the water towards the lowest point of escape causes a considerable lateral movement in the circulation, and this lateral movement has probably, at least in part, controlled the shape of certain deposits on the range which have their longer dimensions parallel to the strike of the layers of the iron-formation.

The ponding of the water and consequent overflow has still another effect. Where ponded the flow is governed by the point of lowest escape rather than by the shape of the impervious basement. When water is drawn off at the edge of a basin, the flow is greatest near the point of escape and diminishes in all directions away from that point. This statement is true, whether the bottom of the basin is flat or fluted; hence, in the Mesabi iron-formation, where the water is ponded, the flow is concentrated near the point of lowest escape regardless of whether this be over a syncline or anticline so far as both are below water-level. The lowest point of escape is likely to be over synclines, but the surface erosion, both by glacial and meteoric agencies, has been such that this is not always the case. For this reason it is not certain that iron-ore deposits near the edge of the Virginia slate or near the edge of interstratified slate-layers may not have developed along arches as well as in synclines of the iron-formation.

The above facts are intimately related to the problem of finding ore under the solid black Virginia slate. The question is frequently asked, is there any reason why ore shall not be found under the black slate? The absence of ore under the slate has not been demonstrated by actual drilling; only a comparatively few holes have penetrated any considerable thickness of the Virginia slate and entered the iron-formation below. Yet such holes as have been put down have revealed

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ore only near the slate-margin and frequently of low-grade. In several cases, the iron-formation beneath the slate has been shown to be of a green, unaltered variety, indicating that the alteration necessary for the development of ore-deposits has not gone far. If the development of the ore is dependent upon a vigorous circulation, and this vigorous circulation is lacking under the Virginia slate because of the ponding, we have here an adequate cause for the non-existence of ore-deposits under the black slate. Yet, further work may show that other factors have entered, and, considering the extent and value of the new iron-bearing territory which would be thrown open were ore found under the Virginia slate, more actual drilling seems advisable to settle the question.

In the Gogebic district, while the impervious basement has been the controlling factor in the concentration of waters, and consequently of the ores, faulting through the dikes has afforded free passages for waters which have been taken advantage of, with the result that the ores follow such faulting-planes and are not uniformly confined to positions on the impervious troughs.

In Figs. 1, 2, 3 and 4, it will be noted that, for all of the districts except the Mesabi, the vertical element in the distribution of the ores is an important one. In the Mesabi district the horizontal element is the greatest one in most cases. Here, a single ore-body or a group of ore-deposits may give practically a continuous surface of iron-ore for several miles, with a depth ranging from a few feet to 400 ft. or more. In the other ranges a depth of ore of 1,000 ft. is common and 2,000 ft. has been exceptionally reached. It is not unlikely that in some places, particularly the Gogebic district, the ores will be found to greater depth, although the lower limit is well determined for the most part. Theoretically, the lower limit of the ore-bodies ought to be the lower limit of the active circulation of oxidizing waters from the surface. In the Mesabi district the proportion of the area of the ore-bodies appearing at the rock-surface to that of the iron-formation as a whole is about 8 per cent. for the productive part of the district and 5 per cent. for the entire district. For the other districts of the Lake Superior region, the area of the iron-ore deposits is far less than this percentage of the area of the iron-formation as a whole. In bulk, the percentage of ores to the iron-formation is much smaller

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throughout the Lake Superior country, for it will be remembered that the ores are essentially surface alterations and are well represented at the surface.

The alteration of the iron-formation, resulting in the concentration of the iron-ores or in the development of ferruginous cherts and jaspers and amphibole-schists, has taken place in different geological periods under various conditions, with a result that the ores of each of the districts, as well as of different parts of the same district, show considerable lithological variety. The same ingredients—iron oxide and silica—appear here mechanically combined as highly crystalline jasper, there as a soft ferruginous chert, or both mechanically and chemically combined as an amphibole-magnetite schist. The iron-oxide may appear here as a brilliant specular hematite or magnetite, and there as a soft granular hematite or limonite. The ores of the Mesabi district are soft and granular and associated with ferruginous cherts. At the east end they become amphibolitic, magnetitic and non-productive. The ores of the Gogebic district are of a similar nature and become amphibolitic, magnetitic and non-productive at both the east and west ends of the district. The ores of the Vermilion district are hard, blue and red ores, at Ely, brecciated, associated with jaspers. The ores of the Marquette district comprise hard blue ores and brilliant specular ores associated with jaspers, called "hard-ore jaspers," soft ores associated with ferruginous cherts, or "soft-ore jaspers," and with underlying slates, and at the west end of the district magnetite and specular hematite ores associated with jaspers and with amphibole-magnetite rocks.

Without going into the variable conditions in the several districts and the varying geological history of the different ores and rocks, it may be said that, so far as the alteration of the iron-formation has proceeded continuously under the influence of surface-waters without interruption by igneous activity or orogenic movements, the soft ores and ferruginous cherts have resulted. So far as these have been subsequently under deep-seated conditions of alteration, they have become dehydrated into hard red and blue specular ores and brilliant jaspers. This phase of the alteration did not require the agency of surface oxidizing-waters. So far as the alteration of the original iron-formation took place within the sphere of influence of great intrusive

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masses where the waters were heated and oxygen was not abundant, or under similar conditions developed by deep burial and orogenic movement, the iron oxide and silica of the formation combined, with small amounts of other substances, to form ferrous silicates, and the ferrous iron was oxidized to magnetite, making one of the variety of rocks usually described as amphibole-magnetite schists.

In the discussion of the development of the ores, we have begun with the original rocks of the iron-formation, the iron carbonates and greenalite rocks and minute amounts of sulphide, that is, the rocks which constituted the formation when it was first deposited. The origin of these rocks themselves is a subject of absorbing interest, but it is a subject in which direct evidence, such as we have been dealing with, is lacking; and a subject, furthermore, of less practical importance and less direct bearing upon the study of the ores themselves. Certain facts are well known. Both the iron carbonates and the greenalites (and their altered equivalents) constitute a conformable part of a continuous sedimentary succession, being interbedded, overlain and underlain by fragmental rocks, such as quartzite and slate. They are themselves bedded. It is impossible to escape the conclusion that they are water-deposited sediments. It is further clear that they are not water-deposited sediments of an ordinary nature. They are not fragmental; their nearest analogues are chemical sediments, such as limestones.

From here on in our explanation we must depend rather upon analogy with chemical sediments, such as limestone, and with iron compounds now being precipitated in bogs and lakes and elsewhere, than upon direct evidence in the formation itself. On this basis the history of the development of these rocks, as outlined by Van Hise,[12] is as follows:

"It is . . . my belief that the iron for the iron-bearing formations was largely derived from the more ancient basic volcanic rocks of the Lake Superior region. When the individual districts are taken up it will be seen that a greenstone, often ellipsoidal, in many places porous and amygdaloidal, in many places

[12] Iron-Ore Deposits of the Lake Superior Region, by C. R. Van Hise, 21st *Annual Report, U. S. Geological Survey*, Pt. III., pp. 305–434.

See also:

Monograph XLIII., U. S. Geological Survey, by C. K. Leith.

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schistose, and rich in iron, is the most characteristic rock of the Archean, and that similar rocks occur abundantly in the Huronian. Where these igneous rocks were adjacent to the seas they would be leached by the underground-water and the iron transported to the adjacent seas. It is probable that to some extent this leaching process also went on below the waters of the sea. The iron was probably transported to the water mainly as carbonate, but to some extent as sulphate. The carbonate would there be thrown down by oxidation and hydration as limonite, and the sulphate in part as basic ferric sulphate. Much of the sulphate was probably directly precipitated as sulphide by the organic material. The limonite would be mingled with the organic matter which was undoubtedly present, as shown by the associated carbonaceous and graphitic shales and slates. When deeply buried the organic matter would reduce the iron sesquioxide to iron protoxide. By the simultaneous decomposition of the organic matter carbon dioxide would be produced, which would unite with much of the protoxide of iron, producing iron carbonate. The sulphate of the basic ferrous sulphate would be reduced to the sulphide by the organic material, thus producing the pyritic carbonates. Where the iron was brought to the water mainly as sulphate the direct reduction of this salt by organic matter would form iron sulphide with little or no carbonate. Simultaneously with the production of these substances chert was formed, probably through the influence of organisms.[13]

"Some of this silica would unite with a part of the protoxide, producing ferrous silicate. More or less mechanical sediment would also be laid down. Thus the original rocks—the cherty iron carbonates, the ferrous silicate rocks, and the pyritic cherts—would be produced.

"It has chanced that at three different periods in the history of the Lake Superior region, these processes of the development of the original rocks of the iron-bearing formations have occurred extensively. While this might at first be thought remarkable, there is no good reason for thus regarding it. At some time during each of the Archean, Lower Huronian and Upper Huronian periods the quiescent conditions of chemical and organic sedimentation have occurred, and since the iron-bearing volcanic rocks were each time available for the work of underground-waters and sea waters, naturally iron carbonate and the other original rocks have been produced. In each period the source of the material and the process of its formation were essentially the same."

Time of Concentration.

The concentration of the ores of the Lake Superior region was far advanced before Cambrian time, as shown by the existence of ore and other iron-formation boulders, derived from the pre-Cambrian, in Cambrian conglomerates. For each of the districts the particular time of concentration has been more closely calculated, but this is a subject which would require a longer discussion than the scope of this paper warrants, and

[13] The Penokee Iron-Bearing Series of Michigan and Wisconsin, by R. D. Irving and C. R. Van Hise: *Monograph, U. S. Geological Survey*, vol. xix., 1892, pp. 246–253.

Fossil Medusæ, by C. D. Walcott: *Monograph, U. S. Geological Survey*, vol. xxx., 1898, pp. 17–21.

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the reader is referred to Van Hise's general paper already cited.

Topographic Relations of the Ores.

With very few exceptions, the ore-deposits of the Lake Superior region lie either on the slopes or at the foot of conspicuous ranges or hills. This has been explained by Van Hise as due to the concentration of ores through the circulation of ground-waters. Topographic elevations have given the waters sufficient head to search the ground on their slopes and perhaps at the base of the slopes. On the slopes the movement of the water is largely downward and more or less direct from the surface, thus carrying an abundance of altering agents, particularly oxygen and carbon dioxide, while in intervening low-lying areas the waters escape with a lateral and upward movement after a longer underground-journey, during which they have lost considerable proportions of the agents altering the iron-formation to ores. He has held that, in the latter positions, the ores have not developed so abundantly as on the slopes. The present topography is in many places not the same in detail as the topography which existed at the time the ores were concentrated, and accordingly it is not safe, in discussing the relations of the ores to the topography, to consider too small topographic units. Believing that the present major topographic conditions represent, at least in part, the past conditions, Van Hise has discussed in some detail the relations of the Lake Superior ores to the particular topographic features of the different districts. In one or two cases, and especially in the Gogebic district, the topographic units selected for discussion may have been too small, and this has resulted in criticism of the entire theory. It is believed that his main conclusions as to the relations of the ores to the major topographic features have been confirmed rather than disproved by recent work.

Present Studies of the Origin of the Ores.

The ores of the Lake Superior region have been shown in certain districts to have developed from the alteration of iron carbonate, in other districts from the alteration of iron silicate, and in other districts from both. In the Marquette, Gogebic, Vermilion, and Crystal Falls districts, the original rock has

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been described as iron carbonate. In the Mesabi district, the original rock has been described as consisting of ferrous silicate or "greenalite" granules principally, but with subordinate amounts of iron carbonate. In the Felch Mountain and Republic areas, the presence of granules similar to the Mesabi granules has been noted. In the Menominee district, both iron carbonate and silicate granules have been supposed to yield the ore.

The vast scale on which the alteration of ferrous silicate granules has occurred in the Mesabi district, together with the evidence of the somewhat wide-spread distribution of such granules in unknown but small quantity, has led to a re-examination of the ores and rock of the remaining districts, with the result that evidence of the prior existence of granules has been found also in both the Gogebic and Crystal Falls districts. So far as the re-examination has gone, however, it tends to confirm the essential correctness of the determination of iron carbonate as the original iron-formation rock for these districts.

In the new Animikie iron-range, on the northwest coast of Lake Superior, the eastern continuation of the Mesabi range, original and secondary iron carbonate and pseudomorphs after greenalite are found so well exposed and so closely associated in distribution that it is hoped that the study now in progress of this district may furnish decisive evidence of the real relations of these two substances.

In the new Baraboo iron-range of Wisconsin, which is of pre-Cambrian age and similar in many respects to the Lake Superior deposits, Weidman has attempted to show that the ores are original bog-deposits, which have become dehydrated. His conclusion does not necessarily affect the remaining Lake Superior ores, but the similarity in geology is so close that, until the contrary is proved, it is believed that the explanation applied to one region will apply substantially to others. Reasons are given on a subsequent page for believing that his explanation is not a true one.

Experimental work in the laboratories of the University of Wisconsin show the great ease of the alteration of the iron carbonates through the agency of water. In a few hours this substance becomes coated with hydrous iron oxide when treated with warm water containing only small amounts of oxygen de-

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rived from contact with the air. Iron silicate has been found to be much more refractory, but still has yielded slightly to the action of ordinary water. Quantitative results are looked for at an early date. Analysis of mine-waters for the Lake Superior region as a whole, it is hoped, may yield results significant of the manner of alteration of ores and rocks of the iron-formation.

Investigations show the easy solubility of the phosphorus compounds in the ores. Incidents may be cited of the remarkable change in phosphorus-content in stock-piles and mines, because of the washing of surface-waters, and in drilling, allowances are frequently made for changes in phosporus-content in washing. The form in which the phosphorus occurs is not definitely settled. Some of it is certainly in the form of apatite. But many ores, and especially Mesabi ores containing phosphorus, apparently do not contain apatite. The association of phosphorus with aluminum, both in ores and other phases of the iron-formation, suggests the possibility that the phosphorus is chemically combined in some manner with aluminum. It is hoped that investigations now in progress will settle these questions.

Iron-Ranges of Recent Discovery.

As a result of the vigorous exploration constantly going on in the Lake Superior region, new areas of iron-formation are discovered from time to time. Among those newly-discovered areas now receiving attention several may be particularly noted.

Baraboo Iron-Range.—The Baraboo quartzite-bluffs of south central Wisconsin constitute a pre-Cambrian outlier, surrounded by flat-lying Paleozoic rocks, well to the south of the main area of Lake Superior pre-Cambrian rocks. Within the last few years it has been discovered, largely through the explorations of Mr. W. G. LaRue, that associated with the quartzite and beneath the Paleozoics are iron-formation, slate and dolomite. Iron-ore has been found in the iron-formation and the shipment of ore began in 1903. The district has been described in a Bulletin of the Wisconsin Geological Survey by Dr. Samuel Weidman.[14]

[14] Baraboo Iron-Bearing District, by Samuel Weidman. *Bulletin XIII., Wisconsin Geological and Natural History Survey* (1904).

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The Baraboo quartzite, having an estimated thickness of from 3,000 to 5,000 ft., forms an east-west synclinorium about 29 miles long and ranging in width from 2 miles on the east to 10 or 12 miles on the west, resting on a basement of Archean rock. The upturned north and south edges of the quartzite forms respectively the north and south ranges of the Baraboo Bluffs, standing from 700 to 800 ft. above the surrounding country and above the intervening valley. Minor rolls and faults bring the quartzite up at intervals in the valley. Resting conformably on the quartzite in the valley is the Seeley slate with an estimated thickness of from 500 to 800 ft., above this the iron-bearing formation from 400 to 500 ft. thick, bearing the ore-deposits of the district, and above this the Freedom formation, mainly dolomite, with a thickness estimated to be at least 800 feet. Recent drilling at the east end of the valley has shown the probable presence of another pre-Cambrian quartzite formation unconformably overlying the dolomite, iron-formation and slate. Finally flat-lying Paleozoic sediments unconformably overlie the pre-Cambrian rocks and partly fill the valley, and on these, in turn, rests glacial drift.

The main Baraboo pre-Cambrian series seems to be more nearly similar to the Middle Huronian of the Lake Superior region than to any other. The overlying quartzite series may represent the Upper Huronian.

The iron-ore is of low-grade (mainly below 55 per cent.) Bessemer hematite, with soft earthy, hard and black, and banded siliceous phases. A small amount of hydrated hematite or limonite is also present. The rocks immediately associated with the ore, and into which the ore grades, are dolomite, cherty ferruginous dolomite, ferruginous chert, ferruginous slate, and ferruginous dolomitic slate—in fact, all possible gradations and mixtures of the minerals dolomite, hematite, quartz, and such argillaceous minerals as kaolin and chlorite. In the ferruginous rocks associated with the iron-ore, the iron occurs as hematite and also in the form of carbonate, isomorphous with the carbonates of calcium, magnesium and manganese, in the form of ferro-dolomite and manganic-ferro-dolomite, and as silicates combined with various proportions of alumina, lime, magnesia and manganese, as chlorite and mica, and also very probably to a small extent an iron phosphate.

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The iron-ore is stratified and is conformable with the associated stratified rocks, both below and above. The ore-bodies therefore have the dip and strike of the associated rocks, and are found dipping at various angles from nearly horizontal to nearly vertical.

At the Illinois mine, the first to be discovered and opened, 3 miles southwest of North Freedom, on the south limb of the syncline, the ore-deposit has a thickness of from 30 to 35 feet. The dip of the ore-body and adjacent formations is about 50° N. Between the ore and underlying Seeley slate are 100 ft. of alternating beds and thin seams of iron-ore with a considerable thickness of dolomite and ferruginous dolomite, ferruginous chert and ferruginous slate. Between the ore-deposit and the overlying dolomite are alternating strata of similar character grading up into the nearly pure dolomite. North of the Illinois mine a thickness of 600 ft. of dolomite is known to occur.

Underground-exploration seems to show that the essential conditions and character of the rock existing at the Illinois mine prevail at other places in the district where ore has been found. Several such places are along the strike of the Illinois deposit on the south side of the valley, one on the north side of the valley, and one or more adjacent to the quartzite brought up by minor rolls or faulting in the central part of the valley. The ore, thus far found, lies along or adjacent to the eroded edges of the iron-formation. The few attempts to find ore within the deeply covered parts of the iron-formation, far removed from the eroded edges, have met with failure.

Dr. Weidman concludes that the Baraboo ores were originally deposited as bog-deposits and that their present character is due to subsequent dehydration. The evidence favoring this view is summarized by Dr. Weidman as follows:[16]

"1. The iron ore-deposits are bedded and, to all appearances, stratified like other sedimentary deposits.

"2. The stratified iron-ore deposits are not set off sharply from the surrounding associated stratified rocks, such as slate, chert and dolomite, but grade into them through all possible gradations. The iron-ore is not especially associated with any particular kind of the various rocks adjacent, and the stratification of these various kinds of rock is always conformable to that of the iron-ore. Since the slate, dolomite and chert are original sedimentary deposits, and since the iron-

[16] *Op. cit.*

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ore grades into them and is conformable to them, it is believed that the iron-ore has the same origin as these conformable interstratified deposits of related rock.

"3. That the physical conditions in the district at the time the iron-formation was deposited were favorable for the formation of such shallow-water deposits as iron-ore is indicated by the presence of sun cracks in the rocks of the iron-formation immediately adjacent to the iron-ore and also by the presence of carbonaceous material, probably decayed vegetable matter in the iron-ore and associated ferruginous rocks. Furthermore, the rapid alternation in the various strata of the iron-formation indicates changing conditions of sedimentation,—a common characteristic of shallow-water deposits, and the composition of the iron-bearing formation itself, with its chert, carbonate rock and slate, is identical with the composition of shallow-water deposits being formed to-day.

"4. That the iron-ore deposits originated long antecedent to the folding and fracturing of the iron-formation and the deposition of the mineral veins is shown by the fact that the strata of iron-ore are generally folded, crumpled and fractured, and that the mineral veins cut across the stratification and also across the folds of the iron-ore and associated rocks. The folding of the ore-deposits appears to conform in all respects to the folding of the associated rocks, and the ore-deposits appear to have the position and distribution which they should have if formed before the general folding of the pre-Cambrian formations into their present position took place.

"5. The change in the ore subsequent to its original deposition as limonite, as shown by the microscopic and chemical study of the rocks, is believed to be mainly a change to hematite by dehydration of the limonite under deep-seated conditions of metamorphism. This change is exactly parallel to the dehydration of the original clay-minerals of the gray Seeley slate-formation and of the clay-minerals in the slate phases of the iron-formation, which now contain very little water of constitution, only 2 or 3 per cent., but which originally must have contained from 10 to 25 per cent. This change of limonite by dehydration is also analogous to the probable dehydration of the original siliceous deposits now constituting the chert layers.

"The geological data which led the writer to believe that the Baraboo iron-ore was very probably deposited as limonite under conditions similar to those under which bog and lake ore are formed to-day, and later merely partially dehydrated to form hematite, have just been briefly outlined. The principal evidence believed to be directly opposed to the theory of the secondary development of the iron-ore as replacement and alteration deposits by work of the underground-water is mainly furnished by the character of the work of the underground-water at present, as indicated by its chemical composition, and by the work of the ground-water of the past, as indicated by the character and composition of the mineral veins in the ore and associated rocks.

"From the study of the composition of the ground-water now circulating through the iron-bearing rock and associated formations in the district, and its comparison with that of ground-water outside the district and with river waters and with chalybeate mineral waters, it has been concluded that very probably the present work of the ground-water in the iron-formation is not that of depositing iron-ore. It is quite generally accepted that mineral veins are deposited from underground-water circulating through fractures in rocks: and the fact that the mineral veins in the iron-ore and associated rocks are largely quartz and not iron-ore is interpreted as evidence that the work of the ground-water of the past was very probably not that of depositing iron-ore. If the work of circulating ground-water of the past could have or did develop the iron-ore deposits, why should not

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the work be now in progress since ground-water is now circulating through the ore-deposits and associated rocks as it has in the past; and if the iron-ore could have been or was developed by the work or agency of ground-water, why should not the mineral veins that ramify through the ore-deposits and associated iron-bearing rocks be composed largely of iron-ore instead of quartz?'

As the Baraboo district is closely related lithologically and stratigraphically with Lake Superior ore-bearing districts, Weidman's conclusion that the ores are original deposits immediately suggests that the same origin may hold for the ores elsewhere in the Lake Superior region, or, vice versa, and that explanations of the secondary origin of the Lake Superior ores may really apply in the Baraboo district. It is believed that Weidman has not proved the Baraboo ores to be original for the following reasons:

His arguments 1, 2 and 3 without change may be used equally well in support of a theory of secondary concentration from an originally deposited chemical sediment—iron carbonate or iron silicate—which has been described as put down in shallow waters.

No evidence is presented in support of argument 4. The secondary alteration of previously folded and contorted iron-formations to iron-ore, with the retention of all the folded and contorted structures, is the common feature for the Lake Superior country. Accompanying and following this change, there has been some additional folding, followed in some cases by an introduction of normal mineral veins, but usually it has been a minor phenomenon. The fact of the folding and contortion of iron-formation layers could not be accepted elsewhere in the Lake Superior country as evidence of the subsequent occurrence of the major folding.

The argument that the presence of quartz-veins shows that quartz only, and not iron-ore, has been introduced or concentrated subsequent to the deformation, it is believed, will not stand analysis. In the Lake Superior region, abundant evidence may be presented to show that quartz-veins have developed both before and after the concentration of the iron-ore, and I know no criteria that will enable one to say that the presence of quartz, rather than iron-ore, in particular fractures or veins proves or disproves the secondary origin of the ore. In the Vermilion district, minute veins of ore may be seen cross-

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ing both quartz-veins and an earlier concentration of the ore, and in turn are crossed by later quartz-veins.

The dehydration of limonite (5) has unquestionably occurred, but it would make little difference whether the limonite were an original deposit or were itself a secondary alteration.

The argument that the composition of the waters seems to show that ore-concentration is not at present occurring is beside the mark when we remember that for the Lake Superior region as a whole, and hence for the Baraboo district itself until the contrary is proven, the concentration of the ores was largely finished before Cambrian time. The conditions at that time doubtless differed widely from present conditions, but even admitting that they are similar, the fact that waters from the iron-formation, which have been analyzed, contain no ingredients different from the surface-waters of the region, does not show that they may not still be doing work. Van Hise has emphasized the fact that the ordinary surface-waters are the ones that do the work, and that the development of iron oxides from iron carbonates and iron silicates is the normal work of percolating waters for the belt of weathering in general. The presence of silica in the Baraboo waters shows that silica is being carried away from this general zone or region, and there is every reason to believe that it may be carried as well from the ore as from the overlying sandstone. Certainly it could not be assumed that the silica contained in the water, notwithstanding the fact that it is no more abundant in mine-waters than elsewhere, might not be coming from the ores themselves, thus enriching them. It is believed, also, that the process of alteration has gone so far in the mines from which the waters were secured that the waters now circulating there for the most part are in contact with oxidized products—the end-products of alteration under such conditions—and that they could scarcely be expected to contain any considerable amounts of ferrous compounds. It is not at all certain that waters working through less-altered parts of the formation, either in the past or elsewhere in this district, might not show a different composition.

Finally, Weidman's theory would require that, after straightening out the secondary folds and contortions, the iron-ore should lie in a continuous even bed or layer with only such irregularities as would result from uneven deposition in shal-

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low waters. If, in the Lake Superior region as a whole, the iron-formation layers should be straightened out, the resulting bed or layer would have strange shapes indeed, and would here and there end abruptly along the strike or dip against other phases of the iron-formation or even against a dike or boss of greenstone. (See Fig. 1.) Indeed, Weidman's theory is squarely opposed to nearly all conceptions of the structural relations of the ores to the adjacent rocks worked out independently for each of the Lake Superior districts, where exploitation has gone far enough to allow of satisfactory study. It is believed that these structural facts worked out for the region as a whole will apply in the Baraboo district itself, so far as the facts are known in the present state of exploration.

The ores thus far found have been along the eroded edges of the iron-formation. Attempts to locate the ore in the center of the great Baraboo valley beneath the dolomite have met with failure. Straightening out the folds in the iron-ore bodies, and reproducing the original attitude and distribution, the iron-ore deposits would appear, so far as is now known, as a series of lenses, limited by dolomite or feathering-out beneath it, down the dip, in the direction of what is now the center of the valley. On Weidman's theory, the ore might be found anywhere beneath the dolomite and its truncation by an erosion-surface would be purely a matter of accident. It must be admitted that drilling has not gone sufficiently far to prove that the iron-ore lenses uniformly reach the pre-Cambrian erosion-surface at some point, but sufficient drilling has been done to lead certain mining engineers of the district and myself to conclude that the evidence is strongly in favor of this view and that the normal Lake Superior conditions here prevail.

While it is believed for the above reasons that Weidman's conclusion as to the origin of the Baraboo ores is not supported by the evidence he has presented, the fact that the ore grades into and is overlain by dolomite—a unique occurrence for the Lake Superior region—would make it easy to accept an explanation of the origin of the Baraboo ores different from that applied to the remaining Lake Superior ores if sufficient evidence be presented, and such acceptance would not necessarily imply revision of views of the secondary origin of the remaining Lake Superior ores.

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Cuyuna Iron-Range.—Iron-formation, containing ore of possible commercial grade, has been recently found in Crow Wing county of north central Minnesota, where surface indications are slight. The discoverer was Mr. Cuyler Adams, of Duluth. For some years magnetic variations had been known in this general area, and the existence of a quartzite-ledge near Dam lake, south of Kimberly, in Aitken county, 15 miles to the east, had been taken to represent the southern margin of the Mesabi syncline by the Minnesota Survey, Mr. Merriam, Geologist of the U. S. Steel Corporation, and others. I published a sketch showing the hypothetic structure of this part of Minnesota on the basis of this and other exposures in 1903.[17] Combining a careful magnetic survey with a conception of the general structure somewhat as above noted, Mr. Adams began drilling in the area adjacent to Deerwood, which happens to be a drainage-divide. This led, in 1903, to the discovery of a magnetite-ore of low-grade, associated with amphibole magnetite rocks. Further exploration, in 1904, has shown the presence also of hematite-bodies in adjacent areas.

The magnetic lines and drilling indicate the presence of two iron-formation belts, one 15 miles long running from a point a little east of Deerwood in a southwest direction, and another 2 miles long, parallel to the first and 6 miles to the northwest of it. I will shortly publish, elsewhere, a description of this area so far as the facts are known from present drilling. It may be said here that the iron-formation associated with the ore is, for the most part, of the metamorphosed amphibole-magnetite variety, similar to the metamorphosed portions of the iron-formations elsewhere in the Lake Superior region. So far as present information goes the iron-formation of the Cuyuna range might be from its lithology correlated with the metamorphosed Vermilion, metamorphosed Mesabi, or metamorphosed Marquette iron-formation, that is, it might be of Archean, Middle Huronian or Upper Huronian age.

Animikie Iron-Range.—Extending from Gunflint lake along the international boundary northeast to the east end of Thunder bay, on Lake Superior, is the Animikie (Upper Huronian) series, which may be called the eastward continuation of the

[17] *Monograph XLIII., U. S. Geological Survey.*

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Mesabi iron-bearing series. The Animikie series is similar to the Mesabi series in that it consists principally of a great slate-formation underlain by a comparatively thin iron-formation. It differs from the Mesabi series as typically developed in the productive part of the Mesabi range, in lacking a basal quartzite, in containing intrusive silk, and in being in some places more metamorphosed. The presence of iron-formation in the Animikie series was known long before the discovery of the Mesabi range; indeed, the series carrying the ores in the Mesabi range was first called the Animikie series, because of its similiarity to the Animikie series on Thunder bay or Animikie bay on Lake Superior. Exploration for ore of commercial grade and quantity in the part of the Animikie series extending from Gunflint lake to the east end of Thunder bay has long been unsuccessful. Renewed activity in the Loon lake district, about 25 miles east of Port Arthur, has recently again attracted attention to the Animikie series. Here, hematite bands from a few inches to 18 or 20 ft. in thickness and averaging in composition, for the most part, below 50 per cent. of metallic iron, have been found extending over considerable areas. There seems to be more concentration here than anywhere else in the eastern portion of the Animikie series. It may be that with hand-sorting a low-grade of hematite may be shipped from here under the unusually favorable conditions of proximity to Lake Superior water and demand for Canadian ore for Canadian furnaces, but this is yet to be proved.

The geological conditions in this region have warranted close study. Mr. W. N. Smith,[18] of the University of Wisconsin, is now preparing a special report on this region. In the course of his work he has discovered the previously unknown existence of a Lower Huronian sediment in this vicinity, and has found the Keweenawan series, with a great conglomerate at its base, to rest in turn on the eroded edges of each of the different phases of the Animikie series and of the Lower Huronian adjacent, proving conclusively the strong unconformity between the Keweenawan and underlying series. Nowhere in the Lake Superior region can this unconformity be seen to such good ad-

[18] *Mining World*, February 18, 1905.

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vantage as here. He has worked out also the presence of definite horizons in the iron-formation, each of them characterized by iron carbonate or of pseudomorphs of greenalite, or both, from which the ores have developed, and has the data at hand for determining finally the true relations of these substances. Also, he has determined the existence of considerable faulting which is partly responsible for the bending and contortion of the iron-formation layers in this area, and probably also for the unusual concentration of ore, which is found along its brecciated edges.

Other Iron-Ranges in Ontario.—The presence of iron-formation is known in a very considerable number of localities in Ontario outside of the Animikie belt. Lists and descriptions may be found in recent reports of the Ontario Bureau of Mines. The commercial value of these new finds has not yet been shown. Geologically those occurring near Lake Superior belong, almost without exception, to the Keewatin division of the Archean or Basement Complex, that is, they are similar in their occurrence and relations to the Soudan formation of the Vermilion district of Minnesota. Principles developed in the study of that district have been found to be of advantage in exploration for iron-formation in Ontario. The only producing-mine in the Ontario portion of the Lake Superior region is the Helen mine at Michipicoten, but magnetite-ores are known in sufficient quantity to be mined in two localities, in the Moose Mountain range north of Sudbury and in the Atikokan district west of Port Arthur, and perhaps elsewhere.

Exploration of Quartzite Areas.

The discovery of ores in covered areas adjacent to the Baraboo quartzite, together with the conspicuous association of quartzite with iron-formation in the Lake Superior district, has led to much study and some underground exploration of areas adjacent to other known pre-Cambrian quartzites of the Lake Superior region. Quartzite is the most resistant of the Huronian series, and it would be expected that it would stand up in conspicuous exposures, while associated softer formations, such as slate- and iron-formation, might be eroded and covered in the low-lying adjacent areas. The exposure of quartzite near Dam lake, referred to in the discussion of the Cuyuna range,

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has long been supposed to indicate the possible southern edge of the Mesabi series; and drilling has been done on this hypothesis. In the Waterloo area of southern Wisconsin, about 25 miles to the east of the Baraboo district, quartzite-exposures are known in such attitudes as to indicate the possibility of their constituting another east-west synclinorium similar to the Baraboo synclinorium, and nearly along the strike of its axis. Presumably, exploration will in time be conducted there to show whether or not iron-formation is present with the quartzite. Near the Neecedah quartzite, drilling has already been done, without success. The Chippewa and Barron county quartzites of northwestern Wisconsin have no associated iron-formation rocks exposed, but exploration will doubtless be conducted in low-lying adjacent areas to determine certainly their presence or absence. The quartzite near Lakewood in northeastern Wisconsin may perhaps represent the southern extension of the Menominee range. In fact, before the Lake Superior region may be said to be thoroughly explored, practically every isolated known quartzite-area must be investigated.

The fact that quartzite underlies the ore in the Mesabi and Gogebic districts, as well as in the newly-discovered Baraboo district, would naturally lead one to explore first the low-lying ground in the direction of the dip of the quartzite, that is, the horizons stratigraphically above the quartzite. But attention is called to the fact that there are in the Lake Superior region, quartzites of three different ages, the Lower Huronian, Middle Huronian, and Upper Huronian; that correlation of any particular quartzite-area with quartzites of known age is practically impossible on the basis of lithological similarity, and that there are for any quartzite-area several possibilities as to the occurrence and distribution of any iron-formation rocks which might be found adjacent.

Work Yet to be Done.

With the appearance of the final general monograph on Lake Superior geology now in preparation, the plan of the United States Geological Survey, inaugurated 20 years ago, will have been carried out. But it must be apparent from a perusal of the foregoing pages that the study of the geology of the Lake Superior region is still far from completion. The work thus

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far accomplished has outlined the major problems and settled many of the important ones, but doubtless 20 years hence geologists will still be wrestling with problems of importance. A consideration of the results obtained and problems still open, keeping in mind the number of men engaged and the funds available for the work to date, enables one to appreciate the extreme complexity of Lake Superior geology and the painfully slow progress which is possible.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

An Automatic Stock-Line Recorder for Iron Blast-Furnaces.

BY J. E. JOHNSON, JR., LONGDALE, VA.

(Annual Meeting, February, 1905.)

Of the many items of information necessary to the successful management of the blast-furnace, few are more important than knowledge of the location and movement of the stock-line:—whether the furnace is full and has been kept so; whether the stock is settling regularly or slipping; and, if slipping, how often it has slipped, at what times, and how far at each time. To obtain these data, a man is sometimes stationed at the top of a pair of mechanically-filled furnaces, simply to watch and gauge them by hand in the old-fashioned way. A more recent practice is the suspension of test-rods from ropes leading over pulleys to the ground-level, where the ropes are wound on small drums with graduated peripheries and provided with crank-handles, whereby the skip-man or some other person can quickly and conveniently gauge the position of the stock-line in the furnace. I believe the credit for this great improvement belongs to Messrs. McClure & Phillips of Sharon, Pa., who were, as far as I know, the first to apply it, and who have a patent for it. The valuable indications gained by application of the Uehling pyrometer to the down-comer are not to be ignored; but the information thus furnished is somewhat indirect, and there is no scale for the translation of "top-temperatures" into "feet down." Moreover, this apparatus as a whole is delicate, and the cost of its installation and maintenance is high.

If the record furnished by the Uehling pyrometer is not in the most desirable form, the test-rods, on the other hand, give no record at all, except through the memory or notes of a man who, to say the least, is not gauging continuously, and who may omit to report an improperly low stock-line, if it be due to a fault of his own.

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The object of the present paper is to describe an apparatus which, it seems to me, meets all these objections, being simple, durable and inexpensive, and giving, by its continuous and automatic action, accurate information concerning the stock-line for every minute of the day.

This device consists essentially of: (1) a test-rod suspended at its top by a chain, but normally resting on the stock and descending with it except when the bell is open; (2) a simple mechanism for lifting this rod out of the way of the in-coming stock at the latter period; and (3) an attachment for recording its motion on a reduced scale.

Before describing these parts in detail, certain special features of the furnace, to which this apparatus is now attached, should be mentioned.

1. The first is the charger, which has been described and illustrated in the recent paper of Mr. T. F. Witherbee read at the Duluth meeting of the Institute.[1] It should be added, however, that the hopper now used is much deeper than that shown in Mr. Witherbee's paper.

The central off-take pipe for the gas makes it easy to locate the test-rod so that it will hang vertically almost in the center of the furnace, while the special setting of the hopper, in conjunction with the type of bell used, makes any other position less convenient.

2. The furnace plant is supplied with water under a natural head of about 60 ft., which is used to operate the bell, the cylinder being placed in a pit below ground-level, in order both to take full advantage of the head and to prevent freezing. Since it is not practicable to have the center of the cylinder under the end of the bell-lever, the rope running up from the former is carried over a sheave so as to give it the necessary direction. This sheave is supported on a pair of I-beams fastened to the furnace at one end and supported on a pipe-column at the other, about 8 ft. above the ground,—a convenient height for the support of the mechanism and the clock-case of the recorder.

The arrangement of the chain supporting the test-rod is clearly shown in Fig. 1. A pulley about 12 in. in diameter is

[1] *Trans.*, xxxv., ——, Fig. —. (In press.)

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carried in a double bearing, resting on the cap of the central off-take pipe. The chain comes up from the test-rod (which is simply a 1½-in. round iron bar, about 18 ft. long), passes through a hole in the cap and around the pulley supported thereon. Thence it goes to a similar pulley carried on a piece of 4-in. pipe held by a bracket on the side of the furnace and stayed to the filling-plates; and passing over this pulley it turns straight down. A short distance below the second pulley the chain is fastened to a piece of No. 10 wire which runs the rest of the way down. Chain was used at the top, because it was doubtful whether wire or wire-rope would stand the combination of heat and small pulleys. The chain is $\frac{3}{16}$ in. in size, and it has given no trouble whatever, though old and nearly worn out when first installed. Fig. 1 shows about half of the center-pipe at the left, and a portion of the bell-lever at the bottom of the picture. The photograph was not "square," having been taken from a rather precarious position on top of the fillers' house—the only available point.

The construction of the mechanism at the bottom is shown almost completely in Fig. 2, except that the operating-cylinder, being in the pit with the bell-cylinder, is of course out of sight. The wire from the test-rod is wrapped around a narrow drum, or double-flanged pulley, 36 in. in diameter with a 2.5-in. face (not shown in the figure), which is mounted next the end-bearing of a horizontal shaft about 7 ft. long, and carries, near its other end, a pinion about 4.75 in. in diameter. The extreme (left) end of the shaft enters the clock-case, as is described later.

With the pinion meshes a rack a little over 2 ft. long, held in gear with the pinion by a guide-roller at the back and carried from below by an extension of the piston-rod of the operating-cylinder.

The cylinder consists of a piece of 5-in. pipe about 30 in. long, bored out true inside and provided with plain screwed flanges. The piston is solid and has water-grooves. The cylinder is open below, and the top head, instead of a gland, has a rather long boss containing a renewable brass bushing which is a working-fit on the piston-rod.

The valve of the bell-cylinder, operated by a wire rope running to the top of the furnace, is so arranged that when it takes

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the position necessary to allow the bell to be opened, water is turned into the recorder-cylinder and forces the piston down, which, through the medium of the rack and pinion, rotates the shaft carrying the drum and winds up the wire, thus raising the test-rod. The piston goes to the bottom and the test-rod is held at the top as long as the bell is open. When the bell-cylinder valve is reversed, the recorder-cylinder is opened to discharge and the weight of the test-rod, acting with its greater lever-arm, pulls up the piston; the test-rod descending until it rests upon the stock at its new level. From this time on, the test-rod descends with the stock, unwinding the wire from the drum and raising the piston, until the bell is again opened and the whole cycle is repeated. The maximum stroke of the recorder-cylinder is about 2 ft., and the ratio of drum- and pinion-diameters being about 7.5 to 1, the maximum travel of the test-rod is about 15 ft., of which about a foot must be sacrificed in order to insure that the rod shall always be raised above the top of the stack and out of the way of the incoming stock.

The counter-weight clamped to the piston-rod extension seen at the bottom of Fig. 2 keeps the slack out of the wire and takes up all lost motion in the rack and pinion. It also assists the cylinder to raise the test-rod which would be a little too heavy for the cylinder alone, though about the proper size for the purpose.

The action of the recording-instrument is shown in Fig. 3. A clock-case with a 24-hour clock and dial (standard type of the Bristol Company) was procured and a cast-iron support was made for it, which also carries the reduced end of the horizontal main shaft and a short piece of 1-in. shafting, which projects through into the clock-case and carries, at its inner end, a narrow segment of a gear, *A*, which meshes with the worm cut on the reduced end of the drum-shaft.

A pen, *B*, is fastened to the end of the small shaft and plays across the face of the revolving-dial, *C*.

To the rear end of the small shaft is fastened an arm about 4 ft. long (shown at the extreme left of Fig. 2), which serves the double purpose of taking up all back-lash in the worm-gear, and of indicating on an arc of a scale of 1 in. per foot how far down the stock-line is in the furnace, without requiring a close inspection of the dial for the purpose. This has proved to be a

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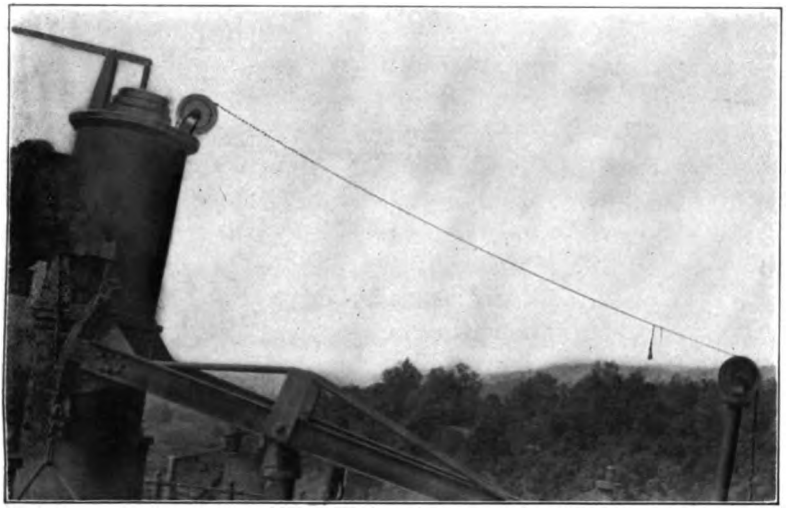

FIG. 1.—THE ARRANGEMENT OF THE CHAIN SUPPORTING THE TEST-ROD.

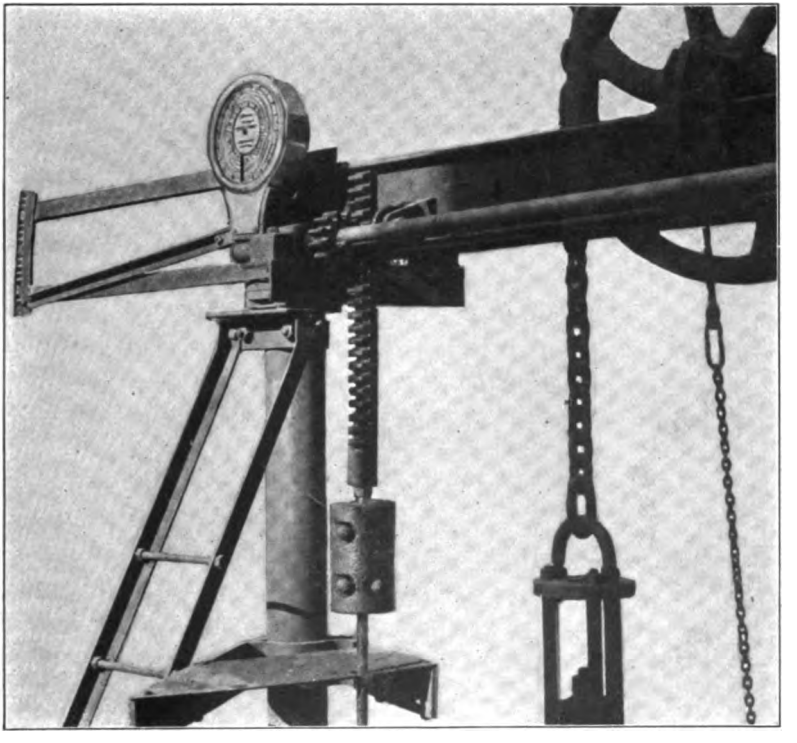

FIG. 2.—THE RECORDING MECHANISM AT THE BOTTOM OF THE FURNACE.

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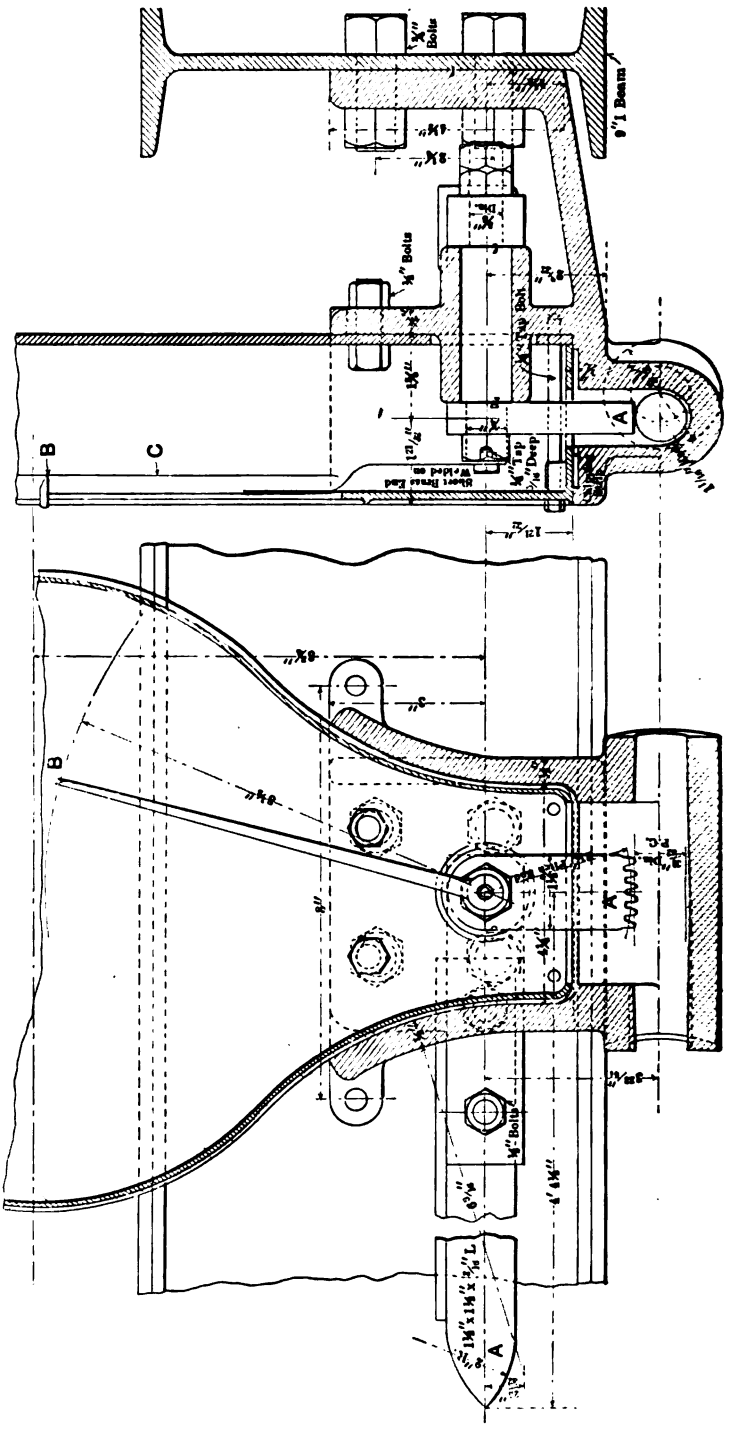

FIG. 3.—BRISTOL'S PRESSURE-GAUGE ADAPTED TO THE AUTOMATIC STOCK-LINE RECORDER.

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rather unnecessary refinement, as the lost motion could have been taken out of the gear by a much shorter lever, and the rack or any part attached to it furnishes a point of attachment for an indicator with a much more open scale and without any additional moving-parts. Moreover, the fillers have attached a tobacco-bag filled with ore to the chain at the top, as shown at the right in Fig. 1, so that it just goes to the outer pulley when the test-rod is at its uppermost position, and, by watching this, they can tell when the furnace is "down" far enough to take another charge. This bag can be seen from all over the furnace-yard and serves as a first-class indicator—which goes to show that one does not necessarily foresee all the possibilities of his own devices, and that the most scientific engineering is not always the most effective.

The character of the charts is best shown by the reproduction of one of them, which is given in Fig. 4. The outer circle, marked "15" (feet), is 10 in. in diameter, and the circles representing feet are 0.155 in., or about one-seventh of an inch, apart; the odd figure being caused by circumstances of the individual case, into which it is not necessary to enter here.

The charts are put on at 10.30 a.m., which is the time for changing shifts, and also cast-time. Four casts per day are made, and the rapid descent of the stock at these periods, followed by the check for shutting-in, are very clearly shown in the record. The extremely regular descent of the stock-line at all other times is also obvious, and will move to envy some of the managers of hard-driven furnaces on Mesabi ores. The chart shown is not at all exceptional in this respect; in fact, if one had been available showing the slipping of the furnace, I should certainly have given it preference, for the sake of illustrating the action of the recorder under those conditions; but, during several months in which the apparatus has been at work, no such record has been made.

It may be explained that the zero-line of the chart has been taken at the height at which the furnace is as full as it can be filled without having the stock interfere with the bell; but the test-rod is raised higher than this, as explained above.

The completeness of the record is more noticeable after a little experience with the device than would be expected from a cursory examination of the chart. It is practically impos-

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sible to "beat" the instrument, and make it show a good record, when a bad one was called for by the circumstances—a fact which the fillers themselves have come to appreciate.

The first night the machine was in operation, the night-shift were late coming out, and knowing that the machine was operated by the opening of the bell, the top-filler opened the bell without waiting to put anything on it, intending to show thereby

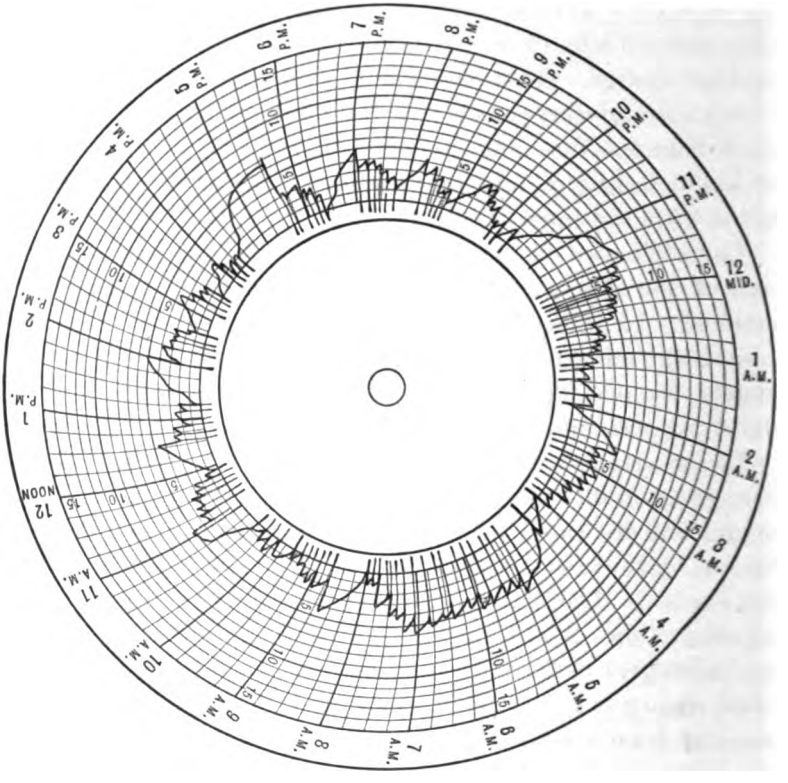

FIG. 4.—CHART FROM STOCK-LINE RECORDER AT FURNACE NO. 2, NOV. 10–11, 1904.

that they had already put up a charge. Of course the test-rod simply rose and descended again to its former level, indicating perfectly what had been done. The admonition delivered the next morning, with an accompanying explanation of the futility of such tricks, has served to prevent their repetition.

So far, there has been no cost for attendance and repairs. The man who oils the hoist- and bell-machinery once a day also puts a little oil on the working-parts of the recorder below.

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(The pulleys on top are not oiled at all, on account of the dust.) The furnace-foreman puts in a new chart, winds the clock and puts ink in the pen once a day; and this is absolutely all the attention the apparatus receives. It was feared at first that, the top of the furnace being rather hotter than good practice permits, trouble might arise with the chain and the test-rod, and also, there being in the gas considerable zinc-oxide vapor, which builds upon whatever it touches, that this would cause trouble by clogging the chain, or perhaps adding to the weight of the test-rod; but these fears have proved groundless.

The circumstances under which the apparatus was installed at the plant described were, as the above statements show, less favorable than the average; yet no difficulty has been encountered hitherto, nor is any anticipated.

In the hope that the device described may be utilized by others, a few points as to its general design may not be amiss.

1. The cylinder should be of ample size to lift the test-rod and still have a good margin of strength; for if, after the falling-back of the stock subsequent to a slip, or for any other cause, the test-rod should be partly buried in the stock, the cylinder should be strong enough to pull it out.

2. The pipe for carrying the operating fluid—steam, air or water—to the cylinder should be small for the size of the cylinder or else well throttled, so that the piston will ordinarily travel at a moderate speed, so as to avoid heavy jerks and inertia-stresses, while, at the same time, if the rod should be stuck in the stock after a slip, when the piston failed to move at first, the pressure would run up rapidly to the point at which it would be able to overcome the resistance and pull the rod out.

3. Some means for cushioning the piston at the end of its stroke should be provided. In the case described, this is accomplished by a helical spring around the piston-rod, which the nut on the latter strikes a few inches above the bottom of its stroke and compresses; but a better method would frequently be to close the lower end of the cylinder and put a small vent for the escape of the air expelled by the downward stroke of the piston at such a point that the piston would overrun it before reaching the end of its stroke and cushion on the entrapped air, very much after the manner of a Corliss "dash-pot," but more slowly.

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4. The cylinder should preferably always be so arranged that the pressure will only be applied during the time that the bell is open, since this interval is short and the tightness of glands, pistons, etc., is of little consequence as compared with those which are under steam most of the time. This can frequently be arranged by connecting the recorder-cylinder direct to one or the other end of the main bell-cylinder, or, in any case, by the use of a three-way valve operated by a connection from the main bell-cylinder valve.

5. Where the bell-cylinder is situated on the side of the stack, or at an inconvenient position for the recorder-dial, the only change required will be to put the cylinder and multiplying-shaft wherever required and lead a wire to transmit the reduced motion of the shaft and test-rod to a convenient position for the clock-dial; the wire operating a lever directly connected to the pen-arm and being kept tight by a weight on its end. This arrangement is indeed simpler than the one here described.

6. The maximum travel of the test-rod is a subject requiring some thought, the natural tendency is to make it as long as possible; but I believe that 20 ft. will be sufficient in most, if not all, cases. If the stock-line has been down more than this through any fault of the fillers, it is sufficient reason for drastic measures without knowing how much further it has been; and if it has slipped more than this, personal attention is demanded, and the added work of gauging by hand is very slight. On the other hand, the size of cylinder, ratio of gearing, and inertia of parts, all increase with increase of stroke; and the increased closeness of the scale on the chart, unnecessary for perhaps 99 per cent. of the time, is a distinct draw-back.

7. One point of minor importance is that the slight jar on the clock-case due to the "bottoming" of the piston has a tendency to loosen the dial on the clock-spindle and may allow it to lag or hasten and thus spoil the accuracy of the time-record. The dial is only held by slight friction, and a very small jar is sufficient to loosen it. The remedy is to have a small thumb-screw through the central boss of the dial, or to have the boss split longitudinally and drawn together by a slightly conical screw-collar milled outside, the tightening of which causes the dial to have a firmer grip on the clock-spindle.

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Several types of arrangement will probably be developed for the apparatus, and some of its details may be modified past recognition, if it should be as extensively applied as its usefulness seems to warrant; but in the great majority of cases, the local design will be a matter of small difficulty if the foregoing suggestions are followed.

POSTSCRIPT.—The device described in the foregoing paper is the subject of U. S. Patent, No. 677,665, dated July 2, 1901, and issued to me. In my judgment, however, it would not be right to omit here a recognition of the issue to Mr. David Baker, May 14, 1901, of U. S. Patent, No. 674,112, for a stock-line indicator, operated by the lowering of the charging-bell. This patent contains no suggestion of a recording-attachment, and does not cover the use of an independent cylinder to operate the test-rod; so that my apparatus, in its present form, does not infringe in any way upon Mr. Baker's invention. This legal question, of course, is not to be discussed in the *Transactions* of the Institute; and I desire here only to acknowledge Mr. Baker's work, and to say that the first knowledge I had of the existence of his patent was after my own had been allowed. Whether, to this day, he has heard of mine, I cannot say. Evidently, we were working at the same time, he in Chicago, and I in Pittsburg, upon the same problem.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Gas-Producer Power-Plants.*

BY SAMUEL S. WYER, COLUMBUS, OHIO.

(Annual Meeting, February, 1905.)

THE installation of the gas-producer power-plant in America has been so unusual that all engineers have viewed it with interest; a large majority, however, regard it with a lack of confidence and many with positive distrust. Despite the fact that European engineers have usually been less inclined to take the initiative along experimental lines than are Americans, they have, nevertheless, developed the gas-producer plant to a very high state of efficiency to which they were forced by the necessity of economy in fuel-consumption.

The gas-producer power-plant is so common in Europe that engineers as well as the general public regard it with the same degree of confidence that is now universally placed in steam-plants. Gas-engines, both small and large, are in general use there, and central stations, aggregating several thousand horse-power, are quite numerous.

The fact that gas-producer power-plants have received so little attention in America may be attributed to five conditions:—1, ignorance and prejudice; 2, newness of work; 3, in-

* By arrangement with the author, the material of this paper, together with that of the two papers, Bibliography of Gas-Producers, and the Testing of Gas-Producers, have been taken for publication from his complete monograph entitled *Gas-Producers*, which is to be published by the Engineering and Mining Journal Co., New York. In this monograph the following subjects are treated:—1. Status of producer-gas. 2. Classification of producer-gas. 3. Manufacture of producer-gas. 4. Efficiency of gas-producers. 5. Use of steam in gas-producers. 6. Effects of CO_2. 7. Thermal calculations. 8. Combustion of producer-gas. 9. Heat-balance of the producer. 10. Fuel for gas-producers. 11. History of gas-producers. 12. American pressure-producers. 13. American suction-producers. 14. English types of gas-producers. 15. French types of gas-producers. 16. German types of gas-producers. 17. Scandinavian types of gas-producers. 18. Gas-washers and purifyers. 19. By-product gas-producers. 20. Gas-producer power-plants. 21. Hints on operation of producers. 22. Testing gas-producers. 23. Future of the gas-producer. 24. Reference tables. 25. Chronological record of early experiments. 26. Bibliography.

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adaptability of gas-engines; 4, fuel-economy not imperative; 5, smoke nuisance not given attention.

1. *Ignorance and Prejudice.*—The only literature pertaining to gas-producer power-plants is that found in the various technical journals and in the transactions of engineering and other technical societies. In many cases the papers are of a fragmentary character, and seldom are they complete or comprehensive. It may be that the lack of reliable data available to engineers is the cause of the ignorance and prejudice that exists concerning this important branch of engineering.

2. *Newness of Work.*—The manufacture of producer-gas is an old process, and gas-engines have been developed to a very high stage of mechanical efficiency, hence there is no valid reason why such installations should be regarded as experimental.

The Winchester Repeating Arms Co., at its plant in New Haven, Conn., has a Loomis-Pettibone gas-producer plant, built primarily to furnish gas for fuel-purposes (such as for annealing-ovens, furnaces, etc.); a 100-h.p. Westinghouse gas-engine was installed some time ago, and later three direct-connected units, each of 175 h.p. have been ordered. At the present time this example is one of the best instances in America of an industrial producer-gas plant where gas is furnished, both for fuel and for power.

The following list comprises some of the larger gas-producer power-plants now in operation in America:

Moctezuma Copper Co., Nacozari, Sonora, Mexico.[1]

The Guggenheim Exploration Co., 700 h.p., Santa Barbara, Chihuahua, Mexico.

Detroit Copper Mining Co., 1,000 h.p., Morenci, Ariz.

Rockland Electric Co., 1,000 h.p., Hillburn, N. Y.

Potosina Electric Co., 600 h.p., San Luis Potosi, Mexico.

Velardeña, Mining and Smelting Co., 2,000 h.p., Velardeña, Durango, Mexico.

The Sayles Bleacheries, 250 h.p., Saylersville, R. I.

It is obvious that much has already been accomplished in this important field of power generation.

3. *Inadaptability of Gas-Engines.*—No gas-producer power-plant can be successful unless the gas-engine is adapted to suit

[1] See *Trans.*, xxxiv., 748–776.

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the particular gas available for its use. On the authority of Westinghouse, Church, Kerr & Co. an engine, which will develop 100 h.p. with natural gas, will give only about 80 h.p. with producer-gas,—a loss of 20 per cent. With a 200-h.p. engine, this loss would be about 15 per cent. and with sizes above 300 h.p. it would be about 10 per cent. Hence, the obvious necessity of designing the engine to suit for the particular fuel it is to use. Several failures have been made by neglecting this important point.

4. *Fuel-Economy has not been Imperative.*—In the list of plants given above, it will be noticed that most of them are in remote regions where the cost of fuel is high, hence the high economy of the gas-producer plant was necessarily a feature that commended itself.

5. *Smoke Nuisance.*—The laxity of the laws regarding the smoke-nuisance has not made it imperative for manufacturers to give attention to the prevention of smoke. As soon as regulations concerning the smoke-nuisance are enforced, the gas-producer industry will receive a new impetus on account of the easy solution that the gas-producer plant offers for this trouble.

General Data.

The following data relative to the design, operation and maintenance of gas-producer plants are given to show their advantages.

1. *Solution of Smoke Problem.*—A good gas-producer, from the very nature of its construction and operation, does not allow the smoke to escape into the atmosphere; hence the gas-producer itself presents a practical solution for the elimination of the smoke-nuisance. The non-requirement of a chimney means a large saving in the first cost and in the maintenance of a power-plant, and is an additional advantage in plants where the æsthetic features of the design are of importance, for instance, in the case of a municipal power-plant.

2. *Labor.*—The cost of labor required to operate a gas-producer plant is about the same as that required in a steam-plant of similar size. However, during the time that a gas-producer plant is idle, it requires less attention than does a steam-boiler.

In the case of a municipal pumping-station, the labor required to operate the producer-gas plant would be one-half that of

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a similar steam-plant, the gas-plant being operated as follows:—The gas-producers to use coal for supplying the gas to operate a three-cylinder vertical gas-engine direct-connected to a triplex double-acting power-pump. In this case the usual fire-engine will be dispensed with, and, should a fire occur, the requisite pressure will be obtained by pumping directly into the system. For ordinary domestic supply, the pump will deliver the water into a water-tower, from which the mains receive the supply as needed. In every case the maximum quantity of water required during a fire is much larger than the average domestic consumption; hence the pump must be designed for this maximum quantity. As a result, the working of the pump at its full capacity for 6 out of 24 hr. would furnish enough water for the daily domestic consumption; the pump would usually be operated from 7 to 10 a.m. and from 3 to 6 p.m.

A gas-holder of sufficient capacity to run the pump for 30 min. is to be filled before the producers are closed down.

Compressed air is to be used to start the engine, which may be put into motion simply by moving a lever.

The engineer is to live adjacent to the plant so that, when an alarm is sent in to the hose-company, and simultaneously to the engineer's home and to the plant, it would be possible for the engineer to have the pump at work direct into the system by the time the fire-company could reach the fire and make hose-connections.

Since the gas-holder would supply the engine until the producers could be started, the above scheme of operation eliminates the necessity of a night-fireman and the keeping up of at least 70 lb. of steam-pressure in a steam-plant. A similar arrangement could be equally well adapted for fire purposes in connection with large industrial plants.

With regard to the skill required, a producer-gas power-plant does not require any greater skilled labor than does a steam-plant of similar size; however, in some cases, it may require time for men, trained to handle steam apparatus, to become accustomed to gas-engines and gas-producers.

3. *Cost of Installation.*—Two well-known engineering concerns give the following data:

"The cost of gas-power plants, including gas-generating plant and gas-engines, up to 500 h.p., is about 25 per cent. higher than the cost of a steam-plant of simi-

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lar size. Large plants, from 1,000 h.p. upwards, cost about the same as a first-class steam-plant of similar size."

4. *Cost of Repairs.*—The cost of repairs on a gas-producer plant will not exceed that of a boiler-plant.

5. *Use of Cheap Fuels.*—In order that a gas-producer plant shall be commercially successful, it must be able to make, from a low-priced fuel, gas that is sufficiently clean for use in an engine. Bituminous slack is usually the lowest-priced fuel to be had; however, anthracite-culm, or even wood, may be cheaper in some localities. In all cases the percentage of sulphur must be low if the gas is to be used in a gas-engine.

Frequently, the use of a mechanically-washed coal will be economical.

6. *Scrubbing the Gas.*—The only reliable way to remove tar and other hydrocarbons from gas made from soft coal is to have the producer so arranged that the gas comes in close contact with an incandescent mass of carbon. No mechanical means has yet been found to be successful, although several forms of centrifugal apparatus have been tried. For the removal of fine dust-particles, however, centrifugal fans have proved very satisfactory.

7. *Fuel-Economy During Hours of Idleness.*—The stand-by loss of heat is very small, being limited to radiation only; a gas-producer is tightly closed during the time it is not making gas and the entrance of air is thereby prevented. This feature is a marked advantage over a steam-boiler under similar conditions.

8. *Time Required to Start Producers.*—Even after a producer has been idle for several hours, it may be started and can be working at its full capacity within 15 minutes. A gas-holder is generally used in connection with the producer, from which a supply of gas can be taken to start the gas-engine instantly and keep it in operation until the gas-producers are making gas.

9. *Time Required to Stop a Gas-Producer.*—A gas-producer may be stopped instantly by simply shutting off the supply of air and steam.

10. *Composition of the Gas.*—The gas from the gas-producer is quite uniform in composition, and as it usually passes first to a holder before reaching the gas-engine, it becomes thoroughly diffused, thus insuring a still greater uniformity.

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11. *Thermal Efficiency.*—The thermal efficiency of gas-producers is generally about 80 per cent. and in some cases it is even higher than this value.

12. *Automatic Feeding.*—It is much easier to use an automatic feeding-device on a gas-producer than on a steam-boiler, because all producers are placed vertically and the fuel can be dropped into position by gravity. The use of an automatic feed always decreases labor and insures more uniformity in the composition of the gas produced.

13. *Rate of Gasification.*—The rate of gasification in a gas-producer is relative to the character of the coal used. The best rate determined by experience is 12 lb. of coal per sq. ft. of grate-area per hour, although some makers have advised as high as 20 lb. of coal. Experience has also demonstrated that too rapid driving opens a wide door for the admission of adverse gasifying conditions.

14. *Poking the Gas-Producer.*—The amount and frequency of poking a gas-producer will depend on the nature of the fuel and the design of the producer. The mechanical agitation of the fuel-bed (as in the Kitson and Fraser and Talbot producers) eliminates poking entirely. In using bituminous coals, the difficulties of clinker-formations is augmented by the production of coke. The judicious use of a steam-blast and automatic feeding will generally reduce poking to a minimum and, in some cases, will eliminate it entirely. Hand-poking is very laborious for the attendant and usually it will be shirked whenever possible. Gas will usually escape around the poke-holes while the producer is being poked, which will vitiate the air in the producer-room and also effects the regularity of the composition of the gas.

15. *Calorific Value of Producer-Gas.*—The calorific value of producer-gas varies from 125 to 150 B.t.u. per cubic foot.

16. *Economy.*—The generation of one brake horse-power per hour with from 1 to 1.25 lb. of coal or 3 lb. of wood, is very common producer-gas power-plant practice at the present time, and the gas contains at least 80 per cent. of the heat-energy resident in the fuel.

17. *No Loss from Condensation.*—A very important advantage of the producer-gas installation is that the gas does not condense or lose power on its way to the gas-engine. On the

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contrary, the cooler the gas the better it is for the engine. With steam, the condensation is considerable.

18. *Leakage of Gas.*—It is easy to prevent leakage of gas from the piping, owing to the low pressure of the gas (about 2 in. of water); whereas, with steam, there is often much loss and inconvenience on this account.

19. *Saving in Shafting.*—By using isolated engines, a large saving in shafting may be made in many cases. It is not possible to do this in steam-plants and still maintain a good economy.

20. *Floor-Space.*—The floor-space required for gas-holders, gas-producers and auxiliary apparatus is about the same as that required in a steam-plant; the holder, however, need not be placed adjacent to the producers, but at any other convenient place.

21. *Control of Operation.*—A gas-producer plant is under much better control than the average steam-plant, because in the gas-producers the air-supply rate of gasification as well as the fuel-supply can be regulated more easily.

22. *Storing of Heat-Energy.*—One of the most potent advantages of the gas-producer plant compared with the steam-plant is the ability of the former to store the heat-energy in a holder where it may be drawn upon for immediate use. In this way irregularities and fluctuations of load need not effect the regularity of the action of the gas-producer. This condition means an economy of operation and convenience of use that are impossible with any steam-plant.

23. *Dual Use of Gas.*—Another important advantage of the gas-producer power-plant is that, in many cases, the gas may be used both for power and for metallurgical purposes, the same pipes being used to supply engines and furnaces. The plant of the Winchester Repeating Arms Co., at New Haven, Conn., illustrates an installation of this character.

24. *Economy of Water.*—In many cases it is a serious matter to secure a sufficient supply of water for a steam-plant and sometimes, even with an adequate supply, the quality of the water is such that it is entirely unfit for use in a steam-boiler. One of the most annoying difficulties of many steam-plants is the trouble caused by the corrosion and subsequent cleansing of the boilers, together with the maintenance of feed-water purifiers.

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The gas-producer power-plant forms an almost ideal solution for the problem of water-supply. With a producer in normal condition, the consumption of water will not exceed 2 lb. per brake horse-power hour. The water used in cooling the gases in the scrubber may be cooled in a simple tower and used repeatedly.

25. *Operating Isolated Machines.*—There is no difficulty in piping the gas for several thousand feet in order to reach an engine that drives an isolated machine; this often makes it possible to dispense with abnormal lengths of line-shafting and the consequent friction-loss, or other unsatisfactory methods of power-transmission. This condition is especially valuable in places where electrical power is not used.

26. *Range of Sizes.*—Standard gas-producers now range from a few horse power to more than 500 h.p. in size.

27. *Danger from Explosion.*—There is less danger of explosion in a gas-producer plant than there is in connection with a steam-plant; moreover, should an explosion occur, it would be much less violent and destructive than that of a steam-boiler.

28. *Location of Producer-Plant.*—If desired, the gas-producer plant may be placed near the fuel-supply, which, in many cases, would reduce the expense of transportation, the gas being piped to the gas-engines or furnaces where it is to be used. This arrangement, which is impossible with a steam-plant, means a decided saving in favor of the gas-producer installation.

29. *Future Field for Gas-Producers.*—The preceding paragraphs show the many strong advantages of the gas-producer as a power-generator; the large number now in successful operation shows that the experimental stage has been passed and that they have become a formidable competitor of the steam-boiler. The time is not far distant when gas-producer locomotives for railroad service, gas-producer portable engines, and gas-producer power-plants for marine service will be in common use.

The advantages of the gas-producer for each of the above three classes are :—

I. Gas-Producer Locomotives being,—

1. *Smokeless*: (*a*) trains and stations may be kept cleaner; (*b*) tunnels may be passed through with greater safety; (*c*) comfort of passengers will be increased.

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2. *Cinderless*: (*a*) fuel-loss will be decreased; (*b*) comfort of passengers will be increased; (*c*) large fire-losses due to sparks will be eliminated entirely; (*d*) insurance-rates on property adjacent to railroads will be less.

3. *More Economical*: (*a*) in fuel, since the amount used would be less than one-half that used on steam-locomotives; (*b*) in water, since the amount used would be less than one-eighth that used on steam-locomotives; (*c*) in time, since the time required to take fuel and water will be less; (*d*) in labor in firing on account of automatic feed and decreased amount of fuel used; (*e*) in idleness, since stand-by losses are very low; (*f*) in number of fuel- and water-stations required.

4. *Safer*, since the danger of boiler explosions is eliminated.

II. GAS-PRODUCER PORTABLE ENGINES being,—

1. *Smokeless*: (*a*) large fire losses due to sparks will be eliminated entirely; (*b*) insurance-rates on property adjacent to where an engine is used would be less.

2. *More Economical* in (*a*) water; (*b*) fuel; (*c*) labor; (*d*) time required to secure fuel and water.

3. *Safer*, the danger of explosion being eliminated.

III. GAS-PRODUCER POWER-PLANTS FOR MARINE SERVICE being,—

1. *Smokeless*: (*a*) ships may be kept cleaner; (*b*) passengers will have more comfort; (*c*) a battleship could conceal its location more easily.

2. *More Economical* in (*a*) fuel; (*b*) water; (*c*) time required to fuel; (*d*) bunker capacity; (*e*) floor-space; (*f*) apparatus required, since all of the condensing machinery would be dispensed with.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Testing of Gas-Producers.*

BY SAMUEL S. WYER, COLUMBUS, OHIO.

(Annual Meeting, February, 1905.)

THE following description of methods for conducting gas-producer tests is probably the first attempt to give the subject an analytical, thorough and comprehensive treatment.

In some cases where tests have been made, the methods have been so unsystematic and ambiguous that it has been impossible to secure comparable results. To eliminate this difficulty, the following provisional code of gas-producer tests—which has some resemblance to the code of boiler-trials of the American Society of Mechanical Engineers—has been developed.

The primary object in testing a gas-producer is to determine whether the producer is working satisfactorily or, what is the same thing, to see if the efficiency is as high as it should be with the type of producer in question, and also to find out if the composition of the resulting gas is adapted to the work it has to do.

In order that the test shall be of any value, it must be thorough and comprehensive, and must be conducted with skill and care. When so conducted, the test will reveal the economy of the producer; and, by making suitable changes, the efficiency will often be increased very much. As a result of the tests made by Jenkins,[1] the efficiency was raised from 56.2 to 71.2 per cent. This shows the large saving that may frequently be made in the fuel-consumption by studying the results of a careful test. The log of the test, given in Fig. 1, shows that, just as soon as the temperature became regular, the percentage of CO_2 decreased and that of CO increased.

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* Taken for publication from Mr. Wyer's monograph, *Gas-Producers* (see p. 521 of this Bulletin).

[1] *Proceedings of the Institution of Civil Engineers*, vol. cxxiii., p. 328.

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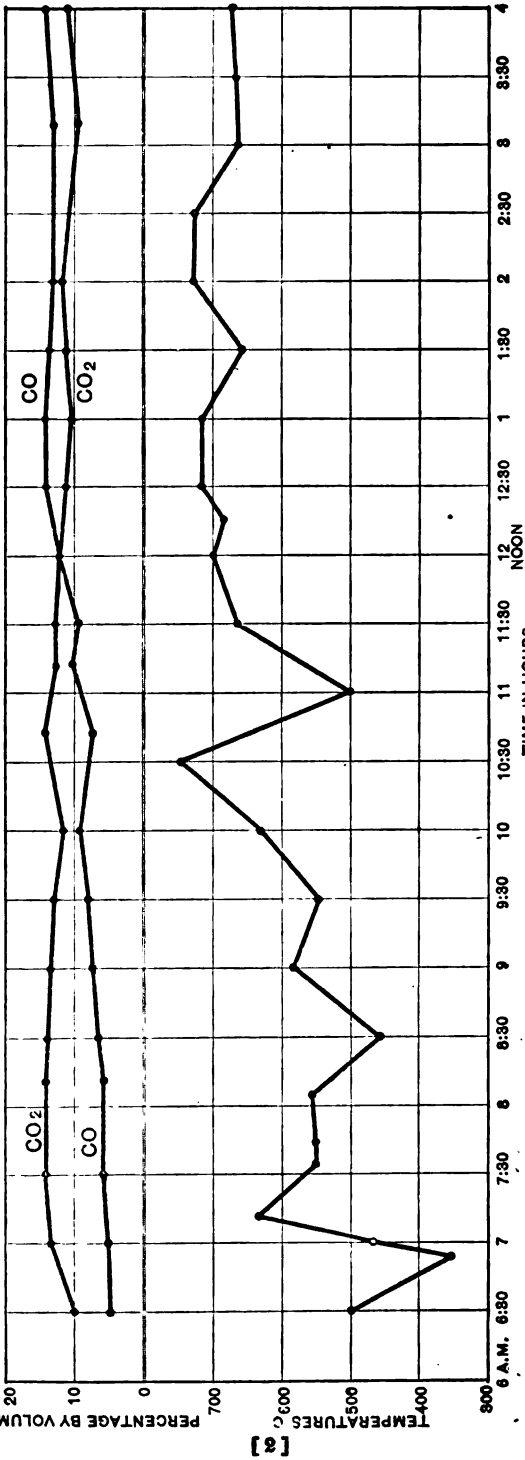

FIG. 1.—CHART SHOWING THE RESULTS OF A GAS-PRODUCER TEST.

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Rules for Conducting Gas-Producer Tests.

1. *Determination of Object.*—Determine at the outset the specific object of making the test,—whether it is to ascertain the capacity of the producer, its efficiency and defects, or the effect of changes of design, proportion or operation,—and prepare for the trial accordingly.

2. *Examination of the Producer.*—Examine the producer in detail, ascertain the dimensions of grates, contour of inner walls, determine the angle of the bosh-wall with the vertical, make a full record describing the same and illustrate special features by sketches. If possible, secure a drawing or make one giving all the general dimensions of the producer.

3. *General Condition of Producer.*—Notice the general condition of the producer and its equipment, and record such facts in relation thereto as bear upon the objects in view.

If the object of the trial is to ascertain the maximum economy of the gas-producer, the producer and all of its appurtenances should be put in first-class condition. Remove clinkers from grates and from the sides of the walls. Remove all dust, soot and ashes from the chambers, gas-connections and flues. Close air-leaks in the masonry and poorly-fitted cleaning-doors. See that all dampers will open wide and also close tight. Test for air-leaks by passing the flame of a candle over cracks in the brick-work.

4. *Character of Fuel.*—Determine the character of the fuel to be used. For tests of the efficiency or capacity of the producer for comparison with other producers, the fuel should, if possible, be of some kind which is commercially regarded as a standard.

5. *Calibration of Apparatus.*—Establish the corrections of all apparatus used in the test for weighing and measuring.

These are:—

(*a*) Scales for weighing coal and ashes and water if an auxiliary boiler is used.

(*b*) Thermometers and pyrometers for taking temperatures; if a thermo-electric pyrometer is used, it must be calibrated with the same lengths of wire and same resistance used in taking the readings.

(*c*) Pressure-gauges, draft-gauges, etc.

(*d*) Apparatus used in making gas-analysis.

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(*e*) Anemometers used in measuring air.

The kind and location of the various pieces of testing-apparatus must be left to the judgment of the person conducting the test; always keep in mind the main object, i.e., to obtain authentic data.

6. *Auxiliary Boiler.*—When an auxiliary boiler is used to furnish the steam for the producer, the amount of fuel it consumes must be charged against the producer. The amount of water that the boiler evaporates must also be measured.

7. *Heating of Producer.*—See that the producer is thoroughly heated to its usual working-temperature before the trial.

8. *Duration of Test.*—For tests made to ascertain either the maximum economy or the minimum capacity of the producer, the duration should be at least 12 hours of continuous running, after the producer has been brought up to its normal working condition.

9. *Starting and Stopping a Test.*—The conditions of the producer in all respects should be as nearly as possible the same at the end of the test as at the beginning. The fire should be the same in quantity and condition, and the walls, flues, etc., should be of the same temperature. In no case should the fires be drawn out, as is often done in boiler-tests. In producers that must be shut down for cleaning, it is advisable that the test should cover one continuous phase only.

10. *Uniformity of Conditions.*—Arrangements should be made to utilize the gas so that the rate of gasification may be constant during the test. Uniformity of conditions should prevail as to the pressure of steam and air-blast, the thickness of fire and bed of ashes, the times of fire and quantity of coal fired at one time, frequency of poking, and the intervals between the times of cleaning the fires.

11. *Keeping the Records.*—Take note of every event connected with the progress of the trial, however unimportant it may appear. Record the time of every occurrence and the time of taking every weight and every observation.

12. *Quantity of Steam.*—When an auxiliary boiler is used for each producer, the amount of steam used can easily be determined from the amount of water evaporated in the boiler.

In the absence of an auxiliary boiler, proceed as follows:—After the test has been made, remove the steam-nozzle and cali-

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brate it by determining the amount of steam that will pass through in a unit of time with the same pressure and percentage of moisture used during the test. Then examine the boiler that is furnishing the supply of steam and determine as accurately as possible the quantity of coal used per hour in making the quantity and quality of steam used per hour, and charge this amount of coal to the producer.

The amount of steam may also be determined as follows:—The hydrogen in the gas and water-vapor must come from three sources, namely, coal, moisture in coal and steam. As all these quantities are known except the latter, it can easily be calculated.

13. *Quality of Steam.*—The percentage of moisture in the steam should be determined near the nozzle where the steam enters the producer, by means of a throttling- or separating-calorimeter. The sampling nozzle should be placed in a vertical steam-pipe.

14. *Measurement of Ashes and Refuse.*—The ashes and refuse will generally be wet before they are drawn from the producer, especially if a producer of the water-seal type is used. After the test, rake out all the ashes and weigh them immediately; in the meantime set aside a sufficient sample for chemical analysis and weigh it; then let this large sample dry in the air until it reaches a constant weight, then reduce to a laboratory sample and determine the residual moisture. The amount of incombustible material should be accurately determined, and, in this way, find the grate-efficiency of the producer.

15. *Sampling the Coal and Determining Its Moisture.*—The same method that is given in the American Society of Mechanical Engineers' code for boiler-trials is to be used, with the following exception. In sizing the sample, crush same to 0.5-in. mesh and air-dry for 24 hours or more,—long enough to insure that the quantity of moisture remaining will vary less than 1 per cent.

16. *Calorific Tests and Analysis of Coal.*—The method adopted by the American Chemical Society is to be used.[2]

17. *Gas Analysis.*—The gas is to be analyzed according to standard chemical methods.

18. *Calorific Value of Gas.*—The calorific value per cubic

[2] *Journal of the American Chemical Society*, vol. xxi., p. 1116 (1899).

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foot should be calculated from its chemical composition and also determined directly by the Junker calorimeter. The two values should correspond closely.

19. *Determination of Water-Vapor, Tar and Soot in the Gas.*— The use of the following apparatus, designed by Prof. N. W. Lord and shown on Fig. 2, is advised.

B is the sampling tube made of 0.5-in. pipe which is placed in the gas-flue; *A* is an annular jacket surrounding *B*, and has pipe-connections at *D* and *C*.

Live steam is blown in at *D*, and out at *C*, the object of this being to keep the temperature of the iron pipe, *B*, below the

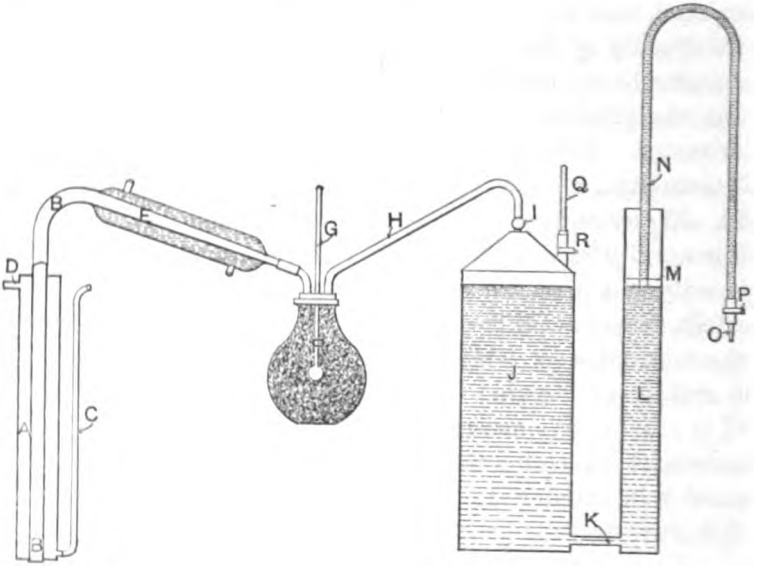

FIG. 2.—APPARATUS FOR SAMPLING GAS, DESIGNED BY PROF. N. W. LORD.

point at which the iron would act on the CO_2. This will secure a sufficient cooling and yet will leave the temperature high enough to prevent the condensation of moisture.

E is an ordinary condenser through which cold water is circulated.

F is a small flask filled with ignited asbestos-fiber and containing a thermometer, *G*.

J and *L* are tanks filled with water and connected at *K*. *I* is a valve. *H* is a rubber tube connecting *J* and *F*. *Q* is a thermometer placed in a stopper in a pipe with valve, *R*, the object of this valve being to make it possible to remove the

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thermometer, when gas is in the tank, *J*. *M* is a float to which is fastened the curved glass tube, *N*, which acts as a siphon and which has a small nozzle, *O*, with a pinch-cock, *P*, on the rubber connection. The object of the float and glass tube is to keep a constant head above the nozzle, and thus insure a uniform flow through it. The operation of the apparatus is as follows:—Disconnect the rubber tube, *H*, and fill the tanks, *J*, and *L*, with water until it overflows at the valve, *I*; fill the siphon, *N*, with water and close the stop-cock, *P*; attach the rubber tube, *H*, with stop-cock, *I*, and circulate water through the condenser, *E*, and steam through the water-jacket, *A*. Then open valve, *P*; the water will be drawn out of tanks, *L* and *J*, and the gas will be drawn through condenser, *E*, flask, *F*, and tube, *H*, into the top of the tank, *J*. The water in excess of the saturation of the gas at the temperature of the small flask is condensed and any tar and soot in the gas retained in the ignited asbestos in the flask. After the test, the flask and its contents are weighed and the increase over the weight taken before the test gives the quantity of the tar and water condensed from the volume of the gas which has passed through the flask. This volume is determined by measuring the quantity of water which had run out of the aspirating-tank, *J*, which had been used in drawing the sample.

The quantity of water remaining in the gas, after passing out of the little flask used as a receiver, is then calculated from the temperature of the issuing gas, which was saturated with water-vapor, by the ordinary saturation tables. The water in the gas is then the sum of the permanent vapor and that condensed. The water in the flask is determined by drying the contents over sulphuric acid to constant weight and determining the loss. The dry contents are then ignited and the further loss of weight estimated as soot and tar.

B = barometric pressure.
Tt = temperature of gas in tank.
Tb = temperature of gas-flask.
Vt = volume of wet gas in tank at temperature Tt.
Vs = Vt reduced to 0° C. and 760 mm.
Vd = volume of dry gas at 0° C. and 760 mm.
Bt = aqueous tension of water-vapor corresponding to Tt.
Bb = aqueous tension of water-vapor corresponding to Tb.

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W = weight of 1 cubic unit of water-vapor corresponding to Tb.

Wb = weight of water-vapor condensed in flask.

Wt = weight of permanent water-vapor in volume Vs.

$$Vs = \frac{Vt\,(B - Bt)}{760\,(1+0.00366\,Tt)},$$

$\frac{Bb}{B}$ = percentage by volume of water-vapor in flask.

$\frac{Bt}{B}$ = percentage by volume of water-vapor in Vs.

$$Vd = Vs\left(1 - \frac{Bt}{B}\right).$$

$Vs\,\frac{Bb}{B}$ = total volume of permanent water-vapor in Vs.

$Vs\,\frac{Bb}{B}\,w = Wt.$

Wt + Wb = total weight of water carried in volume, Vd, of gas.

Thus $Vd = Vt \times \left(1 - \frac{Bt}{760}\right)$ = volume of dry gas.

$Vd \div \left(1 - \frac{Bt}{760}\right)$ = volume of moist gas.

$\left[Vd \div \left(1 - \frac{Bt}{760}\right)\right] \times \frac{Bb}{760}$ = actual volume of water-vapor in volume of gas measured in tank.

Let weight of this be Wt.

Let weight of H_2O collected in bottle be Wb.

Wt + Wb = weight of H_2O to volume of dry gas.

From this, the amount of H_2O and tar and soot per pound of coal can be calculated directly.

19. *Report of Test.*—The data and results should be reported in the manner given in Table I. It is also recommended that the full log of the test be shown graphically by means of a chart, represented by Fig. 1.

TABLE I.—*Data and Results of Gas-Producer Test.*

General Data.

1. Test made by.....
2. Test made to determine.....
.....
3. Chemical analysis made by.....

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4. Type of producer.....
5. Producer built by.....
6. Date of installation.....
7. Kind of fuel.....
8. Form of grate.....
9. Form of ash-pit.....
10. Form of blower.....
11. State of weather.....
12. Barometer in producer room.....
13. Date of test.....
14. Duration of test.....

Dimensions of Producer.

A complete description and drawings of producer should be given on an annexed sheet.

15. Grate-surface..... Width..... Length.....
Diameter..... Area.....
16. Height of bed of ashes.....
17. Height of top of fire above grate.....
18. Thickness of ash-zone.....
19. Position of air-pipes.....
20. Position of steam-inlets.....
21. Diameter of producer.....
22. Inclination of bosh-wall.....

Average Pressures.

23. Steam-pressure near nozzle (lb. per sq. in.).....
24. Force of draft in ash-pit (in. of water).....
25. Force of draft in gas-flue (in. of water).....
26. Steam-pressure in auxiliary boiler (lb. per sq. in.).....

Average Temperatures.

27. Of external air.....
28. Of producer room.....
29. Of steam near nozzle.....
30. Of air entering pre-heater.....
31. Of air entering producer.....
32. (Number of degrees of pre-heating).....
33. Of escaping gases from producer.....
34. Of escaping gases from economizer.....
35. Of ash-pit.....
36. Of ash-zone.....
37. Of combustion-zone.....
38. Of decomposition-zone.....
39. Of distillation-zone.....
40. Of feed-water entering auxiliary boiler.....
41. Of water entering jacket.....
42. Of water leaving jacket.....
43. Of water entering spray.....
44. Of water leaving spray.....

Fuel.

45. Size and condition.....
46. Total weight of fuel fired.....

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47. Percentage of moisture in fuel.....
48. Total weight of dry coal consumed.....
49. Total weight of ashes as drawn out.....
50. Percentage of moisture in ashes.....
51. Total weight of dry ashes.....
52. Percentage of carbon in dry ashes.....
53. Total combustible consumed.....
54. Percentage of incombustible in dry coal.....
55. Total combustible consumed in auxiliary boiler.....
56. Total combustible required to generate the steam used in producer when the producer was used without its own auxiliary boiler.....
57. Total amount of combustible used in the production of the gas.....

Proximate Analysis of Coal.

	Of Coal. Per Cent.	Of Combustible. Per Cent.
58. Fixed carbon.....	.....	.....
59. Volatile matter.....	.....	.....
60. Moisture.....	.....	.....
61. Ash.....	.....	.....
	100 per cent.	100 per cent.

Per Cent.

62. Sulphur, separately determined.....

Ultimate Analysis of Coal.

	Of Coal. Per Cent.	Of Combustible. Per Cent.
63. Carbon (C).....	.....	.....
64. Hydrogen (H).....	.....	.....
65. Oxygen (O).....	.....	.....
66. Nitrogen (N).....	.....	.....
67. Sulphur (S).....	.....	.....
68. Ash.....	.....	.....
	100 per cent.	100 per cent.

Moisture in sample of coal as received.....

Analysis of Ash and Refuse.

69. Carbon.....
70. Earthy matter.....

Consumption of Fuel.

71. Total fuel consumed per hour in running producer.....
72. Total combustible consumed per hour in running producer .....
73. Dry fuel per sq. ft. of grate-surface per hour consumed in producer itself.....

Calorific Value of Fuel.

74. Calorific value by oxygen-calorimeter per pound of dry fuel. .....B. T. U.
75. Calorific value by oxygen-calorimeter per pound of combustible .....B. T. U.

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76. Calorific value by analysis per pound of dry fuel.....B. T. U.
77. Calorific value by analysis per pound of combustible.....
.....B. T. U.

Quality of Steam.

78. Percentage of moisture in steam.....
79. Number of degrees of super-heating.....

Quantity of Steam.

80. Apparent weight of steam per hour.....
81. Actual weight of steam per hour.....
82. Ratio of steam to air-supply.....

Quantity of Air.

83. Percentage of moisture in air.....
84. Apparent weight of air per hour.....
85. Actual weight of air per hour.....

Water.

86. Total weight of water used in jacket.....
87. Number of heat-units carried out per pound of coal.....

Efficiency.

88. Grate-efficiency of producer.....
89. Apparent hot-gas efficiency.....
90. Actual hot-gas efficiency.....
91. Apparent cold-gas efficiency.....
92. Actual cold-gas efficiency.....

Cost of Gasification.

93. Cost of fuel per ton delivered in producer room.....
94. Cost per British Thermal Unit in gas.....

Poking.

95. Method of poking.....
96. Frequency of poking.....

Firing.

97. Method of firing.....
98. Average intervals between firing.....
99. Average amount of fuel charged each time.....

Gas Analysis.

Per Cent.

100. Carbon dioxide (CO_2).....
101. Carbon monoxide (CO).....
102. Oxygen (O) (as admixed air).....
103. Hydrogen (H).....
104. Marsh gas (CH_4).....
105. Olefiant gas (C_2H_4).....
106. Sulphur dioxide (SO_2).....
107. Nitrogen (N) by difference.....

100 per cent.

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108. Pounds moisture in gas per pound of coal.....
109. Pounds soot and tar in gas per pound of coal.....
110. Calorific value of gas from analysis.....
111. Calorific value of gas determined with calorimeter.....
112. Specific heat of gas.....
113. Figure of merit of gas.....
114. Carbon ratio $\frac{C}{H}$.....
115. Volume of gas per pound of coal.....

I wish to express my thanks to Prof. N. W. Lord of the Ohio State University for criticisms and suggestions on this code.

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SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Bibliography of Gas-Producers.

BY SAMUEL S. WYER, COLUMBUS, O.

(Annual Meeting, February, 1905.)

THE following abbreviations have been used in the text:

Cassier's, Cassier's Magazine.

Eng. Lond., The Engineer (London).

Eng. Mag., Engineering Magazine.

Eng. and Min. Jour., Engineering and Mining Journal.

Eng. News, Engineering News.

Eng. Rec., Engineering Record.

Engng., Engineering.

Engr., The Engineer (Chicago).

J. I. and S. I., Journal of the Iron and Steel Institute.

Jour. F. I., Journal of the Franklin Institute.

Jour. Assn. Eng. Soc., Journal of the Association of Engineering Societies.

Mar. Eng., Marine Engineering.

Mem. Soc. Ing. Civ., France, Memoires de la Societe des Ingenieurs Civils, France.

Prac. Eng., Practical Engineer.

Proc. Engs. Soc. of W. Pa., Proceedings of the Engineers' Society of Western Pennsylvania.

Proc. I. C. E., Proceedings of the Institute of Civil Engineers.

Proc. I. M. E., Proceedings Institution of Mechanical Engineers.

Trans. A. I. M. E., Transactions of the American Institute Mining Engineers.

Sci. Am., Scientific American.

Sci. Am. Sup., Scientific American Supplement.

Zeitschr. d. V. D. Ing., Zeitschrift des Vereines Deutscher Ingenieure.

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Eng. Lond., vol. xcviii., August 12th, p. 151. Description and illustration of a Crossley gas-producer plant.

Eng. Lond., vol. xcviii., November 4th, p. 450. Description and illustration, with summary of test of a Pierson suction gas-producer.

Power, January, p. 1. Discussion of the "blast-furnace as a gas-producer;" describes plant and apparatus used in cleaning the gas.

Power, January, p. 50. Brief discussion of the "blast-furnace as a gas-producer."

Power, February, p. 72. Description and illustration of the Crossley gas-producer for bituminous coal.

Power, February, p. 100. Comprehensive and illustrated discussion of gas-power for central stations.

Power, April, p. 201. Brief description of a power-plant where wood is used in the gas-producers.

Power, June, p. 362. Illustration and description of the Bollinckx suction gas-producer.

Les Gazogenes, by Jules Deschamps. This book is divided into 15 chapters, contains 432 pages with 240 figures, and gives a comprehensive discussion of the manufacture and use of producer-gas.

Oesterreichische Zeitschrift fuer Berg und Huettenwesen, September 24th. Discussion and description of the methods used in successfully gasifying peat, where the gas is to be used in gas-engines.

Mar. Eng., October. Description and illustration of a suction gas-producer installed on a ship.

Sci. Am. Sup., September 3d. Illustration and description of a Pierson gas-generating system.

Zeitschr. d. V. D. Ing., November 26th, p. 1793. Extensive discussion of the results obtained with a gas-producer designed for experimental work. Gives valuable tables and diagrams.

Journal of Electricity, December. Illustration and description of a gas-producer power-plant equipped with a Pierson gazogene of special design.

Zeitschr. d. V. D. Ing., October 29th, p. 1656. Discussion of the principles involved in making power-gas; with a summary of the cost of the various forms of fuel-gas.

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Power, December, p. 736. Description and illustrations of suction gas-producers; thorough discussion of details.

Power, December, p. 789. Description and illustration of the Wile gas-producer.

The Electric Club Journal, vol. i., March, p. 65. Lecture on gas-power plants.

Engr., December 15th, p. 821. Illustration and description of producer-gas power-plant.

Engr., October 15th, p. 717. Description and illustration of the Weber suction gas-producer.

Engr., July 1st, p. 450. Description and illustration of the Bollinckx suction gas-producer.

Engr., June 15th, p. 416. Description and illustration of the Otto gas-producer.

Eng. Lond., August 12th. Description and illustration of a gas-producer plant.

Iron Age, August 18th. Description and illustration of the Thesian Centrifugal Gas-Washer.

The Cosmopolitan, December, p. 169. Brief discussion of the adaptability of gas-producers for marine work.

Proc. I. C. E., vol. clviii., p. 309. Brief discussion of the value of gas-producers.

Proc. I. C. E., vol. clviii., p. 320. Description of methods used in testing gas-producers, giving summary of efficiency, gas analysis and water consumption.

Proc. I. C. E., vol. clv., p. 472. Abstract of article on gas-producer plants for heating.

Proceedings of the South African Association of Engineers, vol. i., p. 131. Discussion of the evolution of the modern gas-power plant.

Iron and Coal Trades Review, vol. lxviii., p. 679. Gives comparison of the cost of power with water-gas, Mond gas, coal-gas and electricity.

Iron and Coal Trades Review, vol. lxvii., p. 1559. Description and illustration of Mond gas-plant.

Eng. Lond., vol. xcvii., March 15th, p. 311. Brief description of a German producer-gas plant using brown coal briquettes for fuel.

Eng. Lond., vol. xcvii., April 8th, p. 370. Description and illustration of a Duff producer-gas plant using soft coal.

Producer-Gas, by Sexton. Extensive discussion of the physics and chemistry of the gas-producer.

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Engr., February 15th, p. 137. Description and illustration of the Letombe gas-producer.

Engr., February 1st, p. 107. Illustration and description of the Campbell suction gas-producer.

Engr., February 1st, p. 104. Illustration, with brief description of a Mond gas-producer plant.

Engr., January 16th, p. 89. Description, with illustration of the Fetu De Fize gas-producer.

Engr., January 1st, p. 27. Description and illustration of Crossley suction gas-producer.

Sci. Am. Sup., January 28th. Illustration and brief description of a Pierson producer-gas plant.

Sci. Am. Sup., February 4th, p. 24,320. Illustration and discussion of gas-producers for locomotive work.

Sci. Am., February 4th, p. 98. Brief discussion of gas-driven locomotives and ships.

Eng. News, vol. liii., January 19th, p. 78. Brief description of a Pintsch suction gas-producer.

Eng. Rec., vol. li., January 21st, p. 89. Same as *Eng. News*, vol. liii., January 19th, p. 78.

Power, January, p. 14. Brief discussion of the use of peat as a gas-producer fuel.

Engr., January 1st, p. 18. Extensive and detailed discussion of gas-producer systems; contains illustrations of the American, Baltimore, Nagel, Weber, Wile and Taylor producers.

Gas-Engine, January. Brief description with illustration of the Wile producer.

Eng. News, vol. liii., February 16th, p. 161. Illustration and description of Pintsch suction-producer.

Gas Power, January, p. 3. Illustration and brief description of the Pintsch suction-producer.

Gas Power, January, p. 5. Description and illustration of the Westinghouse producer for soft coal.

Sci. Am., February 18th, p. 139. Brief discussion of the gas-producer, with special reference to power-plants.

Prac. Eng., vol. xxxi., p. 203. Description and illustration of a suction gas-producer installed on a ship.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Effect of Impurities on the Electrical Conductivity of Copper.

BY LAWRENCE ADDICKS.

(Annual Meeting, February, 1905.)

ONE of the properties of copper, which has done much to give it its present prominent place among the useful metals, is its electrical conductivity,—a property which has now become the chief criterion of the value of the commerical product. In pursuance of that distinctively American principle that, "the best is none too good," metal of the highest conductivity is usually called for even where such extreme purity is of no advantage, for instance, in the making of brass and other alloys. The average brass-founder feels nervous unless his copper is so very pure that it shows a conductivity of 99 or 100 per cent., while the character of the zinc used is usually overlooked in his anxiety.

Electrolytic refining has made it possible to produce copper of a very high degree of purity, the metallic impurities averaging but a few thousandths of 1 per cent.; oxygen, usually present in the form of cuprous oxide, bringing the total up to about a tenth of 1 per cent. The published data bearing upon the relationship between chemical purity and electrical conductivity is, however, very scant,[1] nearly all the work that has been done having dealt with alloys carrying considerable quantities of the foreign elements. In this paper I have brought together the results of experiments extending over a period of several years, that have been carried out with the idea of determining the amount of various elements which would lower the conductivity 3 or 4 per cent.—a series of impure coppers rather than alloys.

There are many precautions necessary to keep the conditions

[1] Matthiessen, in *British Association Reports*, 1860–4. Also table of alloys exhibited by Mouchel at the Paris Exposition of 1889, reported by Hering, *Electrical Engineer*, N. Y., ix., p. 344.

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identical in different experiments, since molten copper is very sensitive chemically to its surroundings and the quantity of many substances required to lower its conductivity several per cent., is exceedingly minute. The method of procedure adopted after considerable experimenting was as follows:—

The copper used was in the form of No. 12 B. & S. annealed wire, compacted together and placed in a round Battersea crucible, size H, 500 g. constituting a charge. The impurity was added in one of three ways, depending upon its melting-point,—1. If high, it was dumped in a weighed amount directly into the bottom of the crucible. 2. If comparatively low, some of the copper wire was rolled out into foil and the impurity encased in it, the whole being put at the bottom of the crucible; and 3. If very low (such as phosphorus), it was wrapped in foil as before, but thrust under the surface of the molten copper just before pouring the samples. After the wire was rammed home in the crucible, a layer of broken charcoal about an inch thick was placed on top, the whole filling the crucible. A lid was put on the crucible which was then subjected to a strong blast in a gas-furnace. The sample was poured 25 min. after turning on the blast, which was just sufficient time to melt the charge thoroughly without over-heating it, and not appreciably alter the "pitch" of the copper. No trouble was experienced in getting elements much below their melting-points at this heat to dissolve in the molten copper. Castings were made in a heated iron mold in the shape of slightly tapering cylinders having an average diameter of 0.5 in. and a length of 7 inches. These castings were swaged hot on an anvil to $\frac{3}{8}$ in. in diameter and then drawn cold to No. 12 B. & S. wire. Then, in order to eliminate the effects of the drawing, the wires were annealed electrically by forcing them to carry a current of 110 amperes. The electric method of annealing is a very simple way of accurately duplicating results, the temperature of annealing having a marked influence on the conductivity.[2] The wires were measured for conductivity and then analyzed for copper and the foreign element. The sum of the copper plus impurity was a general check on the analysis and "pitch."

[2] Addicks, *Transactions American Institute of Electrical Engineers*, xxii., p. 695 *et seq.*

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Table I. gives a general summary of the data, the elements being arranged alphabetically. The data are also expressed graphically in Figs. 1 to 16. The alloying elements were obtained in as pure a form as possible and it is not believed that the results are greatly influenced by impurities in them. These were,—proof-silver 1,000 fine; gold in the form of prills from bullion analyses carrying no impurity, but a negligible amount of silver; test-lead; electrolytic iron free from carbon; and sulphur as cupric sulphide. The other elements were of the chemically pure (C. P.) grade obtainable in the market.

TABLE I.—*Summary of Data.*

Element Added.	1st Sample.			2d Sample.			3d Sample.			4th Sample.		
	Added.	Cu.	Cond.	Added.	Cu.	Cond.	Added.	Cu.	Cond.	Added.	Cu.	Cond.
	Per Cent.	Per Ct.	Per Cent.	Per Cent.	Per Ct.	Per Cent.	Per Cent.	Per Ct.	Per Cent.	Per Cent.	Per Ct.	Per Cent.
Aluminum...	0	99.96	99.6	0.006	99.90	98.6	0.109	99.65	66.8	0.739	99.03	43.5
Antimony....	0.007	99.96	99.6	0.022	99.94	97.2	0.047	99.86	95.4	.....	.....	.....
Arsenic.....	0.004	99.96	99.6	0.007	99.95	96.8	0.013	99.94	93.2	0.140	99.82	62.3
Bismuth	0	99.96	99.6	0.028	99.92	99.6	0.045	99.91	99.3	.....	.....	.....
Cadmium.....	0	99.96	99.6	0.062	99.90	99.5	0.113	99.87	99.1	0.427	99.55	96.1
Gold	0	99.96	99·6	0.089	99.86	98.9	0.149	99.84	98.4	0.317	99.64	96.4
Iron.....	0	99.89	100.5	0.042	99.93	96.8	0.046	99.90	92.9	0.068	99.89	89.6
Lead.....	0	99.96	99.6	0.083	99.82	99.1	0.052	99.86	98.7	0.347	99.56	98.3
Oxygen.....	0.020	99.98	100.7	0.050	99.95	101.4	0.100	99.90	100.5	0.200	99.80	98.0
Phosphorus..	0	99.96	99.6	0.080	.....	52.3	.....	.....	.....	.....	.....	.....
Silicon.....	0	99.96	99.6	0.007	99.89	99.4	0.007	99.89	99.1	0.042	99.89	99.0
Silver.....	0.003	99.89	100.5	0.137	99.81	100.0	0.340	99.60	98.3	0.503	99.49	97.9
Sulphur.....	0	99.89	100.5	0.053	99.93	100.0	0.135	99.83	99.0	0.236	99.75	98.9
Tellurium ...	0	99.89	100.5	0.065	99.82	100.4	0.181	99.74	100.2	0.405	99.65	98.7
Tin.....	0	99.96	99.6	0.052	99.85	97.6	0.097	99.85	92.7	0.295	99.61	79.8
Zinc.....	0	99.96	99.6	0.048	99.91	98.3	0.095	99.79	96.3	.....	.....	.....

In examining the results it is natural to group the elements in their order in the periodic system. This is done in Table II. The column headed "factor" gives the ratio of the lowering of the conductivity to the amount of the impurity present. In obtaining this ratio, tangents to the curves in Figs. 1 to 16 have been used, so that the ratios hold strictly true only in the case of an infinitely small lowering. The factor is seen to bear a general relation to the periodic arrangement, decreasing with increasing atomic weight within any one group, though there is no evident relation between one group and another. This factor is of use when examining the analysis of a copper which shows less conductivity, as a means of indicating the probable cause of the trouble. The data do not yield even an approximate constant for molecular lowering of the conductivity—that

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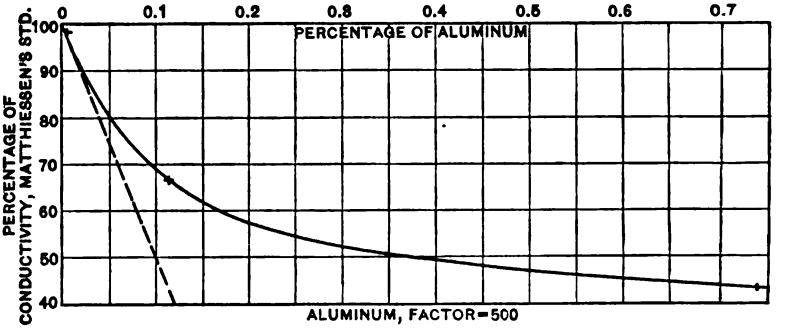
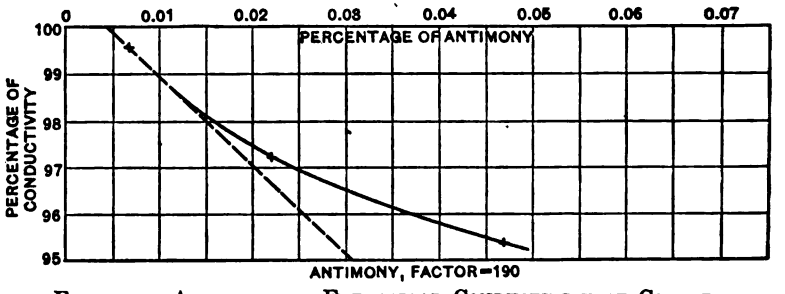
FIG. 1.



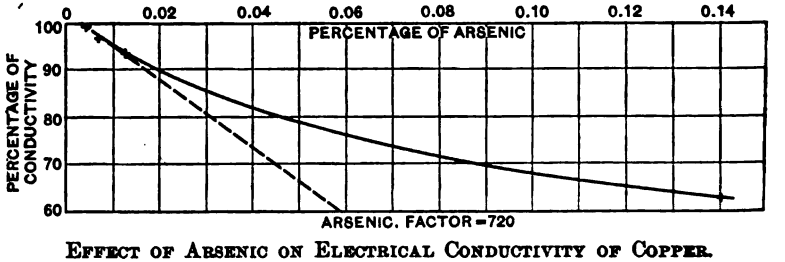
EFFECT OF ALUMINUM ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 2.



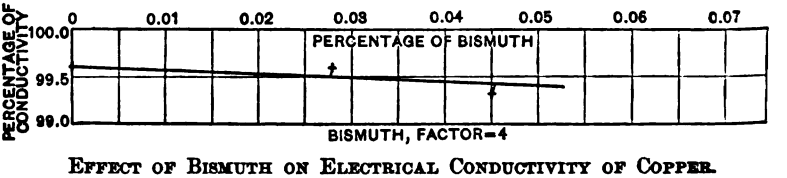
EFFECT OF ANTIMONY ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 3.



EFFECT OF ARSENIC ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 4.



EFFECT OF BISMUTH ON ELECTRICAL CONDUCTIVITY OF COPPER.

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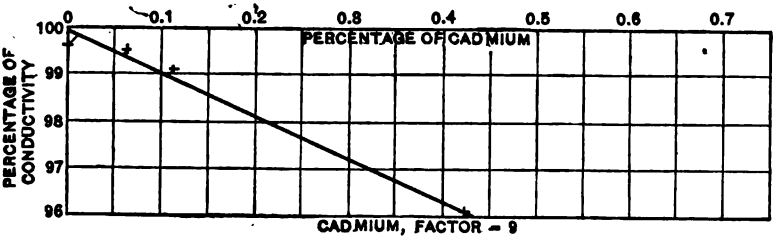
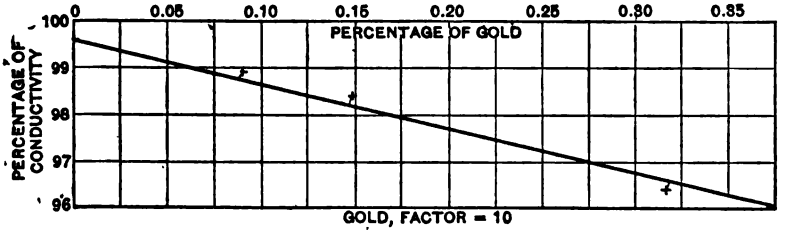
FIG. 5.



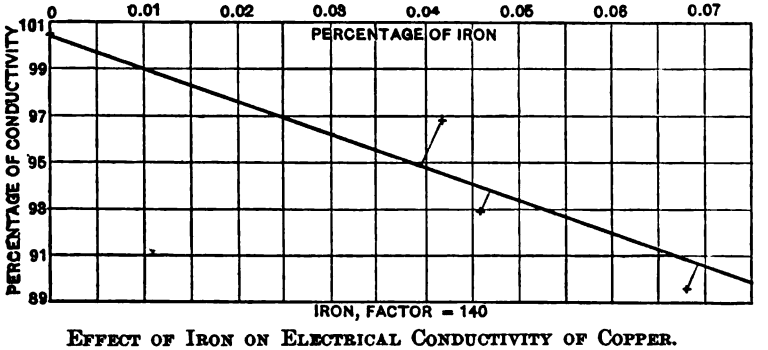
EFFECT OF CADMIUM ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 6.



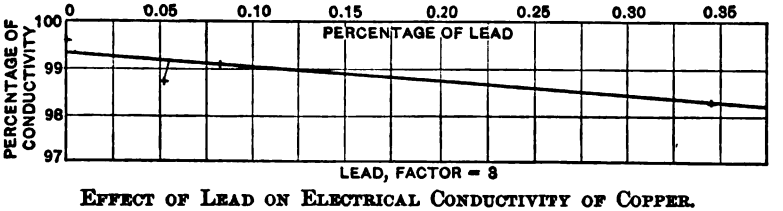
EFFECT OF GOLD ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 7.



EFFECT OF IRON ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 8.



EFFECT OF LEAD ON ELECTRICAL CONDUCTIVITY OF COPPER.

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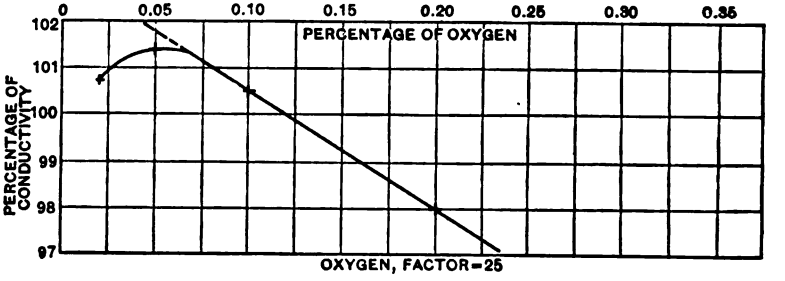
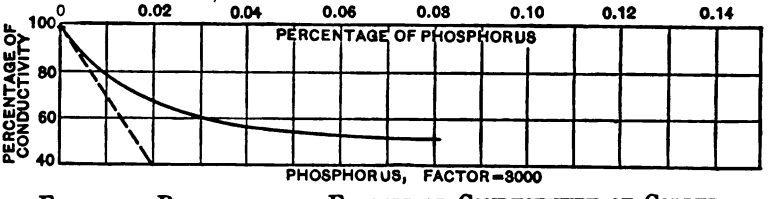
FIG. 9.



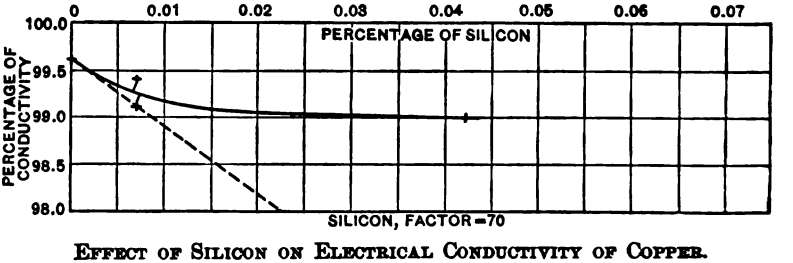
EFFECT OF OXYGEN ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 10.



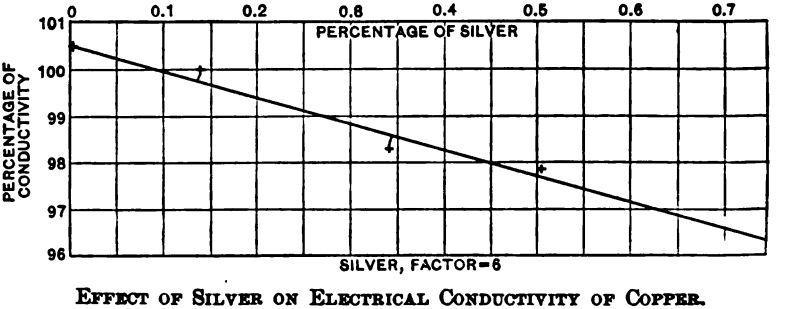
EFFECT OF PHOSPHORUS ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 11.



EFFECT OF SILICON ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 12.



EFFECT OF SILVER ON ELECTRICAL CONDUCTIVITY OF COPPER.

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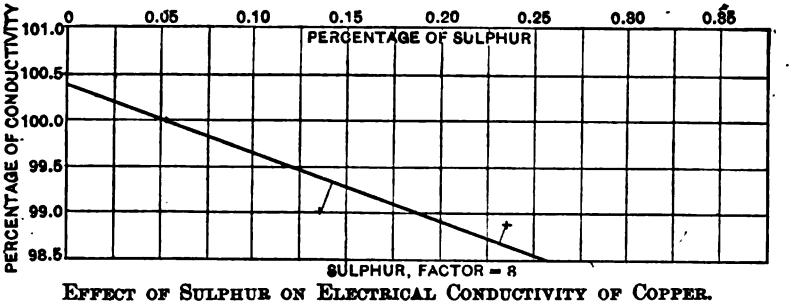
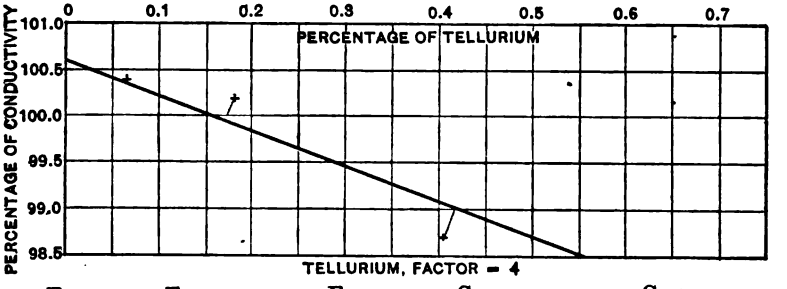
FIG. 13.



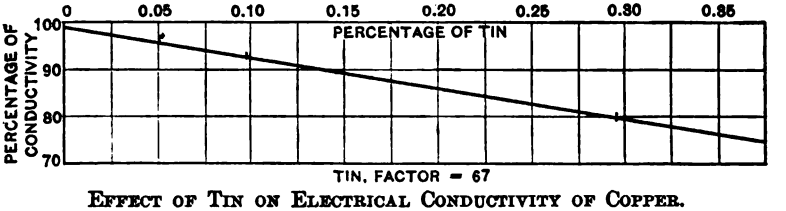
EFFECT OF SULPHUR ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 14.



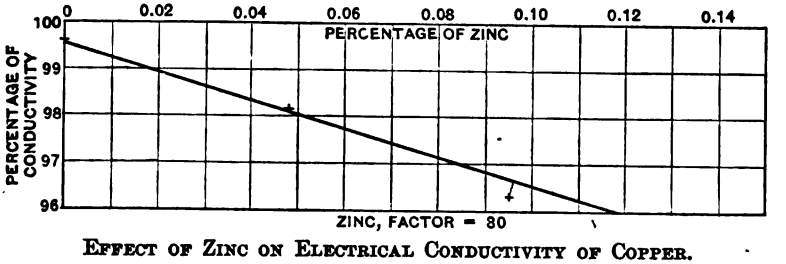
EFFECT OF TELLURIUM ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 15.



EFFECT OF TIN ON ELECTRICAL CONDUCTIVITY OF COPPER.

FIG. 16.



EFFECT OF ZINC ON ELECTRICAL CONDUCTIVITY OF COPPER.

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is, the "factor" multiplied by the atomic weight—even within single periodic groups.[3]

TABLE II.—$Factor = \frac{Percentage\ Lowering\ in\ Conductivity.}{Percentage\ Impurity\ Added.}$

Periodic Group.	Element Added.	Factor.	Atomic Weight.	Remarks.
I	Silver.	6	108	
I	Gold.	10	197	
II	Zinc.	30	65	
II	Cadmium.	9	112	
III	Aluminum.	500	27	
IV	Silicon.	70	28	
IV	Tin.	67	119	
IV	Lead.	3	207	Made metal red short.
V	Phosphorus.	3,000	31	
V	Arsenic.	720	75	
V	Antimony.	190	120	
V	Bismuth.	4	208	Made metal red and cold short.
VI	Oxygen.	25	16	Large amounts made metal brittle.
VI	Sulphur.	8	32	
VI	Tellurium.	4	128	Made metal red and cold short.
VIII	Iron.	140	56	

The form of the composition-conductivity curve of a binary alloy may be one of four different types.[4]

1. When the constituents are mutually soluble in all proportions. The composition-conductivity plot of such a case gives a smooth curve throughout.

2. When the constituents are mutually insoluble in any proportions, in which case the conductivity curve becomes a straight line, the so-called alloy being but a mechanical mixture.

3. When the mutual solubility is limited, there is a soluble series at each end of the curve and a eutectiferous series in the middle. Since two substances absolutely insoluble in one another probably do not exist, case 2 cannot be rigidly applied, such substances forming an extreme application of case 3.

4. Lastly, there are cases where definite chemical compounds are formed which may give a curve with a soluble series at one end and a wholly eutectiferous series at the other.

The results given in this paper deal entirely with the first

[3] Cf. Benedicks, *Zeitschrift fuer Physikalische Chemie*, xl., p. 545.

[4] Howe, *Iron, Steel and Other Alloys.*

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very short part of the conductivity-curve, and it therefore should be expected that some curvature would be shown in every case. Some of the elements employed are so nearly insoluble in copper, however, that the curve differs but little from a straight line connecting the conductivities of the two pure substances in the mixture.

The useful alloys of copper comprise mixtures with zinc, tin, aluminum, silicon and phosphorus. Zinc and tin are added in relatively large quantities to form brasses and bronzes. Aluminum, silicon and phosphorus are added in comparatively small quantities and their ability to deoxidize copper that has been needlessly exposed to the air in melting, probably comes much more frequently into play when making castings than the founder imagines. Under proper conditions it is perfectly possible to make copper-castings of perfect soundness and conductivity, without the use of fluxes. But, looking at the table of factors, it is seen that no worse elements could have been picked out from an electrical standpoint than those above mentioned. On the other hand, the elements which have but little effect upon the electrical conductivity—lead, bismuth and tellurium—make the metal extremely brittle. This effect seems to be a question of solubility, the last-named elements being practically insoluble in copper. We then have a mechanical mixture in which the conductivity is only affected in accordance with the proportions of the mixture, while the mechanical properties of the foreign element itself are brought into prominence.

With the other classes of substances a very different state of affairs exists, the alloy being copper embedded in a matrix of an alloy of copper and the impurity. The conductivity of this matrix is, in general, low, and its amount is beyond all proportion to that of the impurity added, and hence the magnified depression of the conductivity of the whole. Metallographic work with the microscope bears this out, 0.1 per cent. of bismuth, for example, causing but a thin skin around the copper crystals, while the same amount of arsenic forms a thick wall.[6] It is often possible to counteract the detrimental mechanical effects of one impurity by adding another, as in the case where a rela-

[6] Arnold and Jefferson, *Engineering* (London), July 2, 1896.

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tively innocuous impurity may dissolve an otherwise insoluble one, like lead.[6]

The results given in Table I. show what a severe requirement the customary 97- or 98-per cent. conductivity specification is, especially as copper is usually associated with arsenic; and when it is considered that the average electrolytic refinery daily passes this requirement by a margin of 2 or 3 per cent., frequently using anodes containing 1 per cent. or more of arsenic, it will be appreciated that electrolysis would have come as a refining operation even did copper never carry gold or silver.

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[6] Lewis, *Engineering* (London), April 12, 1903.

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The Application of Dry-Air Blast to the Manufacture of Iron.

Discussions of the paper of Mr. Gayley read by title at the Lake Superior Meeting, but first presented at the New York meeting of the Iron and Steel Institute (October, 1904). These papers are here published under a mutual agreement between the Councils of the two Institutes; they will also be found in the *Journal of the Iron and Steel Institute* for the year 1904, vol. ii.

MR. E. WINDSOR RICHARDS, Past-President,[1] said that in 1799 Mr. Dawson, of Low Moor, read a paper before a scientific society in York pointing out the great difference in the moisture of the air going into the blast-furnace in the hot months of the year and in the winter months. A cold blast-furnace was a very sensitive instrument, and showed this very clearly, for in June, July and August the furnace did not drive so well; it did not make the quality of iron and required more fuel; so that Mr. Gayley was on the right road to effect an enormous improvement in this direction. He considered that Mr. Gayley was on the way to great success, and hoped that the apparatus would not be a very expensive one. He supposed it was too early for Mr. Gayley to let them know what the saving was likely to be. If they could produce pig-iron considerably cheaper than was done at the present time it would be a great advantage to the trade, which was at present—on the other side of the water—in a somewhat unremunerative condition. He did not know whether Mr. Gayley would be able to give some idea of the cost of this apparatus or what it would cost per ton, and what the saving was likely to be. That information would be a very great benefit to those interested in the production of pig-iron. Experiments had been made upon desiccation of air for blast-furnace purposes, but had hitherto failed owing to the enormous quantity of air to be desiccated. It seemed to him that it had been left to Mr. Gayley to solve this most difficult problem, and he congratulated him and the Institute upon the valuable paper which he had contributed.

[1] Iron and Steel Institute.

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Dr. R. W. Raymond, Honorary Member:[2] To my mind this paper emphasizes two propositions. The first is, that great aggregations of capital may permit investigations and experiments too expensive to be undertaken by smaller establishments, and (what is equally, if not more, important) may save the time, labor and cost which would otherwise be expended in parallel investigations, separately and even secretly conducted, each of which may fail of useful result, by reason of the lack of adequate support for conclusive tests upon a working scale.

The second proposition is, that great corporations, if they would justify their existence before the present and the coming generation, should make sure that their technical managers are both ready and able to discern the merits and encourage the development of inventions which will, by decreasing the cost or improving the quality of their products, confer lasting benefit upon the world.

Returning to the paper itself, I would call attention to what I regard as one of its most useful suggestions, namely, that the daily variation in atmospheric conditions is really more important in the running of a blast-furnace than the difference in such respects between summer and winter. In other words, we could prepare for more or less continuous conditions of any kind, even the most adverse. It would not be difficult, for instance, to operate a blast-furnace on the Equator, taking the blast from an atmosphere of unvarying temperature, and completely saturated with moisture. Having once determined the necessary fuel-consumption, etc., we could accept the cost thereof as inevitable, and regard the results with equanimity, as the best that could be expected under the circumstances. Similarly, if atmospheric conditions varied only with the seasons, we could prepare for expected changes and endure them with resignation. Indeed, if the variation of atmospheric moisture were confined to changes of season, it may be seriously doubted whether Mr. Gayley's apparatus would effect a sufficient saving to warrant the expense of its installation and maintenance.

But Mr. Gayley's exhibit of diurnal variations presents a

[2] Iron and Steel Institute.

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consideration which, I must confess, seems to me much more important, even financially, and certainly so, when considered from the standpoint of a vigilant and anxious furnace-manager. For it was in my time, and I do not doubt that it still is, the sudden and inexplicable whims of the furnace—its changes without notice, and upon the same burden, blast and blast-temperature, which made the greatest economic trouble. We did not realize that the hygrometric variations of the air from hour to hour could account for such sudden changes; but in the light of Mr. Gayley's experiments, I think this may have been always the chief, if not the only, cause. The secret of successful furnace-management is like the secret of longevity. It is notorious that any list of centenarians will be found to represent all kinds of regimen: Early rising, late rising; total abstinence, steady drinking; tobacco, no tobacco; meat, no meat; hot baths, cold baths; sedentary indoor life, vigorous outdoor exercise, etc. The only element they possess in common is regularity and continuity of practice. That seems to be an invariable element in long life, whatever else may co-operate with it. The man who changes his habits after he reaches fifty is likely to die young! And since longevity is biologically an expression for a maximum of power and minimum cost of repairs, I think the analogy I have suggested is not fanciful, but real, and emphasizes the corresponding principle in our (otherwise totally different) department. In my judgment, the hint in this direction afforded by Mr. Gayley's paper is not its least valuable feature.

Mr. E. H. Saniter (Rotherham) said that the subject was an extremely important one, but he did not propose to deal with the main point, but rather a side-issue, as he was not a blast-furnace man himself. Incidental reference was made to the Bessemer converter. He was informed in some plants when running with only half a per cent. of silicon sufficient heat was obtained. If they took the moisture out of the air, would they be able to make metal low enough in silicon to blow at a proper temperature? Dr. Raymond hit the nail on the head when he said that what was wanted was uniformity of the air, whether it was dry or moist. It might be necessary rather to reverse Mr. Gayley's process and to add moisture by some means so as to make the air uniform, rather than to desiccate it. The only

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other point was with reference to its application to the open-hearth process. That might also be very useful, but they would have to treat the greater sinner—the producer-gas, because it varied more in moisture than the atmosphere. Therefore they would have to deal both with the gas and with the air. In that connection it again came to a question of uniformity. If the Mond gas, which was now used for steel-making, were taken, it would be very regular, although not very dry, and that would be a decided advantage. The drier the gas the better the heat, and in this connection the use of Mónd gas was a distinct gain if sufficiently cooled.

Correspondence.

Mr. W. J. Foster (Darlaston) sent the following communication: The question of the desiccation of that portion of the atmosphere which is necessary for the oxidation of the carbon in the hearth of the blast-furnace by means of refrigeration, will no doubt attract the special attention of blast-furnace managers in the future, especially in those places where the air required per unit of iron made is large, as compared with some furnaces which are working on more modern lines. It is quite evident that the plant that will be necessary for the desiccation of the great quantity of air required for the production of pig-iron will necessarily be of an elaborate nature, depending to a great extent on the cooling surface of the refrigerating chamber, and the particular method of insulation, just as a boiler depends on the heating or cooling surface, assuming that the fall in degrees of temperature in both cases are the same; but, of course, in the case of refrigerating the atmosphere the difference in the temperature of the ingoing air and the temperature of the refrigerating chamber is not so great as would be the case when dealing with the combustible gases in boiler practice, hence a corresponding reduction in its efficiency per unit of tube area. These conditions are based on the fact that the tubes will not conduct or neutralize the heat so rapidly with bodies at nearly the same temperature as would be the case with a great difference in the temperature of the matter inside and outside the tubes respectively. These circumstances will, of course, necessitate the erection of a plant of an elaborate nature; nevertheless, I am of opinion that the refrigerating

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system is by far the best method up to the present suggested for eliminating the aqueous vapor from the atmosphere.

Previous writers on the hot-blast theory tell us that the increased coke consumption which is due to the water or aqueous vapor entering the furnace is equivalent to the direct proportion of the heat necessary to decompose the water molecule. By referring to my paper of May, 1904, on the efficiency of the blast-furnace it will be seen that when dealing with the hot-blast question and the cooling effect of the aqueous vapor in the circumstances that were existing at the Darlaston No. 1 furnace the calculations were simply based on the heat necessary to decompose the water molecule, minus the heat which is given to the aqueous vapor due to heating the blast, only represented a saving of 0.478 cwt. of coke per ton of iron, with a total coke consumption of 29.2 cwt.

A careful examination of my remarks on this subject will show that the above quantity of coke is not the quantity that would actually be saved, but, on the other hand, it is clearly shown that by effecting a saving in this one item by increasing the temperature of the blast we would also have a saving in every other branch of the system, with the exception of the parts clearly defined, the total of which only represents 6.646 cwt. of coke, this being a quantity that is absolutely necessary in these parts of the process which it represents, and the remaining items which correspond to 22.544 cwt. of coke would be indirectly effected by either the superheating of the blast, or any method that would have an influence on the partial removal of the aqueous vapor or its decomposition before entering the furnace. The abnormally high fuel-consumption represented above is chiefly due to smelting silicates of iron with low-blast temperature, etc., and was instanced specially to illustrate the hot-blast theory: The results are very different when working with other materials.

The furnace using the largest quantity of air per unit of iron made, such as is the case in cold-blast practice, will undoubtedly have proportionately a greater advantage of the use of dry air than those using a less quantity. As previously mentioned, the saving will not be found in direct proportion to the air used, but will be found to realize very much greater advantages.

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It is quite certain that the adoption of any practice that will prevent water or water vapor entering the furnace at the hearth will, without doubt, be found to have a great advantage in furnace practice, under the conditions under which they worked at Darlaston. I have estimated that the quantity of heat units necessary to heat the blast to 454.4° C. is equivalent to 2.075 cwt. of coke (measured by the simple complete combustion of carbon and consequently based on the previously recognized hot-blast theory), and the heat necessary to decompose the water vapor was found to be 0.478 cwt. of coke, but in practice I find that this would be equal to at least 10 cwt. in the case of heating the blast to the temperature mentioned, etc., consequently the saving of fuel by removing the whole of the moisture would be represented as follows:

$$2.075 : 0.478 :: 10 :: 2.336$$

which shows a saving of 2.336 cwt. of coke per ton of pig-iron produced. It will be seen that, taking the figures mentioned above, and assuming that the whole of the moisture is eliminated, also putting the average cost of fuel at 14s., which is equivalent to 1.61 shillings per ton of iron, and allowing 20 per cent. for working expenses and depreciation of plant, etc., estimated on 600 tons of pig-iron per week, the approximate saving in round numbers would represent a dividend of 10 per cent. per annum on a capital outlay of £20,400. From the data describing the practical results in the author's paper it appears that Mr. Gayley has solved a very difficult problem.

PROFESSOR H. M. HOWE (New York) wrote that the value of Mr. Gayley's invention was so great that one could hardly rate it justly without danger of seeming theatrical. When we considered the greatness of the pig-iron industry, even the minor advantages of the process, such as the saving of fine ore, the raising of the phosphorus-limit of the ore, and the better control over the sulphur- and silicon-contents of the pig-iron collectively, were of enormous value; but passing these by, the fuel-economy reported fairly took one's breath. Assuming that the world's annual output of pig-iron was, roughly, 46,000,000 tons, and that it called for some 46,000,000 tons of coke, representing some 66,000,000 tons of coal, Mr. Gayley's saving of 20 per cent. as shown by his Table VII. would correspond.

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if applied to all the furnaces of the world, to no less than 13,000,000 tons of coal per annum, or more than twice the total annual coal production of New South Wales, and more than half that of such important coal-producers as Belgium and Russia. We hardly expect this invention to be used at every furnace at home and abroad; yet when we remembered to how great a proportion of all the furnaces of the world Neilson's invention of the hot-blast had been applied, there was very little fear that any large fraction of the world's furnaces would fail to adopt the dry-blast sooner or later.

His remark in the last paragraph but one, that the dry-blast plant "started without a hitch and no difficulties have been developed in any direction," would not greatly surprise those familiar with his exploits.

Refrigerating processes evidently had a great future. It was not necessary to consider such obvious cases as cooling dwellings and factories in hot climates, though failure to do this would, to the future historian, stamp this age as among the barbarous ones. But there was a use which was of interest to those who speculated as to the future of the iron industry. Not a few at this meeting had said with Burns—

> "And forward, though I canna see,
> I guess and fear."

At first sight it certainly might seem that the supply of iron-ore, finite as it was, and not reproduced like our animal and vegetable supplies, would exhaust itself, not indeed in our time nor that of our children, but long before the other great staple materials of industry, such as wood, the textile substances animal and vegetable, breadstuffs, meat and other foods. Yet, on reflection, this idea lost its force; for if the earth really was an enormous meteor with but a relatively thin crust of rocks, there was a supply of iron which would last long after the earth had ceased to be habitable. We depended upon the forests and green fields for our supply of oxygen and of food; but when sun and earth should have so far cooled that the ice-caps spreading out from either pole finally met at the equator; when Mother Earth, exhausted, drew together those icy curtains for her endless sleep, where would be the fields and forests to give us the breath of life? Would not our

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atmosphere then be one of nitrogen plus carbonic acid, and the earth a desert?

But how were we to attack and reach this iron nucleus of the earth? Clearly through these refrigerating processes. When man should have exhausted the limited, and hence exhaustible, deposits of the earth's crust, knowing the vast central mass of iron beneath him, he would be forced to find a means of freezing his way to it.

MR. V. PENDRED (London) wrote: Mr. Gayley appears to have assumed, on very inadequate grounds, that moisture in the atmosphere plays an important part in the performance of a blast-furnace. How he has arrived at his conclusions I am unable to say. To simplify matters and in the end save space, I give here the only passage in the paper which sets forth the reasons which led him to make the interesting and valuable experiment which he describes. "The desiccation of the air used in blast-furnaces in such a way as to reduce its moisture to a small quantity, and to keep it uniform, must of necessity contribute in a very marked degree toward the attainment of uniformity in the furnace operations. The advantages from desiccation can be appreciated only after due consideration is given to the volume of air that is consumed per minute, and the large amount of moisture which it contains. Managers of blast-furnaces are familiar with the chilling effects produced in the hearth by a tuyere that is leaking, which immediately results in a deterioration in the grade of the iron; yet the quantity of water ordinarily entering the furnace under these conditions is not greatly in excess of the quantity carried in, like a steady stream, by the atmosphere, during a period of the average humid conditions prevailing in the summer season in this country." Now I hold that although Mr. Gayley's results are satisfactory, the reasons which he gives for the success which he has attained are entirely wrong. It is evident that he has taken it for granted that water is carried into the blast-furnace with the blast. This might happen provided the blast was cold; but as a matter of fact it was heated for the Isabella furnace to somewhere about 800 degrees F., at which temperature, and some six or eight pounds above atmospheric pressure, water could not exist as such. Under normal conditions damp air is delivered to the stoves, not to the furnace. The

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moisture is at once converted by the red-hot bricks or pipes into superheated steam. The quantity is very small. It remains for Mr. Gayley to show what part it can play in the extremely complex reactions going on in the blast-furnace. On this aspect of the case he is entirely silent. Furthermore, it may be pointed out that the ore, as it comes from the bins, is always damp or even wet, and that the weight of water introduced in this way into the furnace is out of all proportion greater than that of the superheated steam-gas, which goes in with the blast. So far my contention is negative. That is to say, I hold that the moisture in the air could have no appreciable effect on the make of iron, either chemically or thermodynamically.

I now wish to show why the output of the Isabella Furnace was so much increased by refrigerating the air. Good was done, not by drying the air, but by augmenting its density by cooling it. Let us suppose for the sake of argument that, the pressure remaining at 14.7 pounds and the temperature 62 degrees F., one pound of air has a volume of only 6.57 cubic feet instead of the normal 13.14 cubic feet. Then, the blowing-engine remaining unaltered as to size, and running at the same speed, it will deliver per minute twice the weight of air. The work done by the engine will be the same. All the ports and passages will still be adequate. But the rate at which the furnace works depends not on the volume of air blown into it, but on the weight of oxygen passing the tuyeres every minute. Under the new conditions, everything else remaining unaltered, we should double the output of the blowing-engines, and, under suitable charging and tapping conditions, that of the furnace. Unfortunately, hitherto the iron-master could not increase the weight of his blast nor augment the weight of oxygen passing the tuyeres without running his blowing-engines faster. Mr. Gayley has shown us, however, how a very important advantage may be gained in this direction. Let us suppose that he was blowing, normally, at 70 degrees F., that being the inhaling temperature of the blast-tubs. Then each pound of air would have a volume of 13.342 cubic feet. If now, by refrigeration, he reduced the inhalation temperature to 40 degrees, then each pound of air could have a volume of 12.586 cubic feet only, and the weight of air sent into the

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stoves under the two conditions would—the blast-engine running unaltered—vary in the inverse ratio of the two pressures. In other words, the efficiency of the blowing apparatus would be increased by, say, 6 per cent. But we know that Mr. Gayley was on many occasions able to work with much larger temperature differences than these. Thus the inhalation temperature has been reduced from 82 degrees to 23 degrees, at which in round numbers the volume per pound are to each other about as 13.6 is to 12. In this case the efficiency of the blast-engines was augmented by about 13 per cent. This is the reason why Mr. Gayley had to reduce the number of revolutions. Indeed, he has recognized the truth, although he does not attach due importance to it, for he says in one part of his paper, "When the dry-blast was supplied to the furnace it became necessary to reduce the revolutions of the blowing-engines, since the air supplied to the engines was lower in temperature than the natural atmosphere and contained more oxygen per cubic foot, and the tendency of the furnace was to drive too fast."[3] Nor is it to be supposed that the advantage gained was represented wholly by the percentage I have given. Any one familiar with blast-furnace working knows that an augmentation in blast density produces effects larger in proportion than could be anticipated. I have not thought it necessary to go into any detailed calculations. I hope I have said enough to make my meaning clear. The gain derived from refrigeration would have remained about the same if the water had not been removed at all, provided, of course, that the stoves were of sufficient power. At the Isabella furnaces they appear to have been deficient, and the augmented blast density helped them on the one hand, while they were spared the loss of a little energy in converting water into superheated steam. Finally, I can assure Mr. Gayley that I have read his paper with much pleasure, and regard it as a most valuable contribution to our Journal.

MR. B. H. THWAITE (London) wrote as follows: Regarding the characteristic defects of modern blast-furnaces, the supremacy of control over the character of the operations and products of a blast-furnace has been advanced by the important success of Mr. James Gayley's enterprise and ingenuity.

[3] See p. 768.

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It has often been claimed for the blast-furnace that as an instrument of industry it has attained a rare degree of perfection and efficiency, but in recent years it has been demonstrated that a wide margin still exists for further improvements, and especially in methods of thermal utilization. The margin has, however, been greatly reduced by the harnessing of the gas-engine to the blast-furnace. Again, the hot-blast stove efficiency is far too variable, and the contributor, along with others, is attempting to remove this element of thermal inefficiency and irregularity. An ideal blast-furnace operation would be one in which all the agents required for the production, of iron are perfectly controllable in character, in application, and in proportion. But we know that the hygrometric condition of the air is constantly changing, and this variation, added to the vagaries of the thermal output of the hot-blast stoves, is responsible for many of the annoying irregularities in the working of the furnaces. Close observers of blast-furnace operation phenomena have long ago recognized, in perhaps an imperfect measure, the extent of the influence of the hygrometrical or water content variation of the air on the working character of the blast-furnace, but it remained for Mr. Gayley to measure more or less exactly the influence, and at the same time discover a practical means of regulating and reducing the proportion of water introduced in the air-blast into the furnace hearth.

In the period of inception of his refrigerating air-drying system, Mr. Gayley would no doubt be told that the loss of thermal energy involved in his system would probably more than counterbalance the advantages to be derived in reducing to a fixed degree the moisture in the atmospheric air required for furnace operations. To his great credit, Mr. Gayley has had the courage to balance the thermal loss and gain by an actual test on a large and expensive scale, and the balance appears to have splendily justified the test.

The introduction of water into the furnace hearth, whether in the form of aqueous vapor or directly from tuyere leakage, has the following chemical sequence when in contact with iron. The equation is $4H_2O + 3Fe = Fe_3O_4 + 8H$; in contact with the carbon of the fuel the equation is $H_2O + C = CO + 2H$.

The absorption of thermal energy in the dissociation of H_2O

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reduces the furnace hearth temperature; and, although the re-oxidation of the hydrogen in the hot-blast stoves and in the steam-boiler furnaces restores part of this thermal loss, there is always in the use of hydrogen the irrecoverable loss of the latent heat of steam formation—unless methods of condensation, or temperature reduction below 212° F., are available, which for stove and furnace gases they are not.

According to Mr. Gayley, the operation of the refrigerating machinery involves the expenditure of 535 indicated horse-power.

The writer demonstrated, many years ago, that if the blast-furnace gas was used directly for the production of the power, the output would be increased some 400 per cent.; but, assuming only a factor of 300 per cent., there would be an excess of power available by the displacement of the three steam blowing-engines of 671 indicated horse-power; so that the question of additional power supply need not influence the decision as to the wisdom or otherwise of adopting the dry-air process once the complete adoption of gas power is also agreed upon.

In the power expenditure data supplied by Mr. Gayley there is perhaps some miscalculation; the mystery is, however, partly cleared up by the reduction in the resistance of the furnace burden and the reduced volume of air—represented by the fall of the blast-pressure by 8.8 per cent.

The calorific value of the blast-furnace gas of 22.3 per cent. of CO in using dry air is reduced by 2.4 per cent. or to 19.9 per cent. in the absence of the hydrogen data; the calorific value of the gas cannot be determined, but it will probably not exceed 75 to 80 B.t.u. per cubic foot. This gas, both for stoves lighting from the cold or for steam raising, will be very deficient in igniting qualities, and it would be interesting to hear whether this gas is used for steam raising, and whether any auxiliary fuel is required to sustain ignition. The gas is quite suitable for use in specially designed gas-engines, and its calorific reduction could be still further extended without risk of destroying its power-producing usefulness.

On the basis of Mr. Gayley's figures representing the economy to be secured by the use of dry air, and assuming the variation of moisture in the atmosphere between the winter and summer months to be within the limits indicated in the

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Advantages and Disadvantages of the Dry-Air System.

Debit.	Credit.	Credit Balance.
Thermal loss in compressing air.	Greater density of refrigerated air involves less blowing-engine power to provide same weight of air-blast.	Twenty per cent. greater output of iron.
Ditto in lowering temperature of air some 53° Fahr.	Less resistance to flow of gases through furnace burden, owing to more equable and uniform working conditions.	Twenty per cent. reduction in fuel consumed per unit of pig-iron and output.
Reduced calorific value of gas.	Greater control over furnace operations and its product, especially in proportion of silicon.	
Reduced sensitiveness of gas to ignition.	Reduced iron-ore waste.	Reduced iron-ore waste and greater control over silicon range.
Capital expenditure in refrigerating plant and cost of supervision and stoves relating thereto.		

graphic diagram, Fig. 1, it is possible to calculate the economy that should be obtained in a furnace, released from an environment of steam issuing from blowing-engine pump and hoisting-engine and shunting locomotive exhaust-pipes, and the escaping steam from steam-boiler safety-valves, all changes resulting from the supersession of steam by gas power. It may also be assumed that the tuyere water is cooled, without adding to the air moisture-content, and that the chilling of the pigs does not involve any material production of steam or water vapors. Taking the average proportion of moisture in the winter months of November, December, January and February to be equal to 2.15 grains per cubic foot, this proportion, compared with that of summer months' average, is only 73 per cent. less than the average content of refrigerated air in Mr. Gayley's reported tests. Now the average increase in pig-iron output as the result of using dry air equals 20 per cent., and, curiously enough, the reduction in fuel (coke) is represented by the same figure. Therefore, furnaces working in the four winter months named, and in a completely steamless environment, and all other conditions being equal, should give an increased efficiency represented by 14.6 per cent. In other words, the pig-iron output in November, December, January and Feb-

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ruary should be 14.6 greater, and the fuel consumption per ton of pig-iron output should be 14.6 per cent. less than the results of the four summer months of June, July, August and September. There is in these possible winter improvements in output and fuel-economy an adequate *raison d'etre* for the supersession of steam by gas power, a technical reform that would add to the advantages claimed for the dry-air blast, because—

1st. The evils of escaping steam into the atmosphere would be removed; and

2nd. There would be an ample supply of power available for air compression and air refrigerator services.

MR. GAYLEY, in reply, said: With reference to one question that had been asked, as to the changing of the stoves, he would state that there had been no change in their operation. The furnace at the Isabella Works, having four stoves to the furnace, was to have one stove on blast and three on gas, changing stoves every hour. Mr. Saniter had said that he understood, with respect to their Bessemer practice, that his firm was hoping to run with 0.5 per cent. of silicon; no doubt this could be done if the dry-blast were used at the Bessemer department, and by such use a further economy in coke could be obtained at the blast-furnace, but in their practice at the present time, with the use of mixer metal, the silicon-content was from 1 to 1.25 per cent. Mr. Saniter was quite correct in calling attention to the importance to the open-hearth furnace of extracting the moisture from the producer-gas. He (Mr. Gayley) was confident that if the moisture were extracted from both gas and air for use in the open-hearth furnace, and from the air supplied to the converter, it would result in increased product and greater uniformity in the metal. A question had been asked by Mr. Windsor Richards as to the saving obtained by this process. That, of course, varied with each locality, but he had given data from which the economy for any one district could be calculated. It had been found, as a result of several tests, that the saving in horse-power in the blowing-engine room practically compensated for the expense of operating the dry-blast plant. They had found through the use of this process a saving in coke, labor, supplies, and general expenses, in the limestone corresponding to the coke saved, which would otherwise have been needed for fluxing the ash, a reduction of phos-

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phorus in the metal corresponding to the reduction in coke consumption, which was of some importance in the making of Bessemer pig, and, where fine ores are used, there was less waste of ore through furnace gases. In addition to this there was increased earning capacity for the plant through increased output.

In preparing this paper he had only been able, by reason of the limited time permitted in its preparation, to present a description of the process with the results obtained therefrom, while there were many questions relating to the economy effected which would have to be considered later. In the meanwhile it had transpired that certain statements made in the paper had been misunderstood, and he took this opportunity to shed further light, if possible, on some of them. First, with regard to the density of the blast; some erroneous calculations might be made from the data showing the air entering the refrigerating chamber at about 80 degrees and leaving it at about 25 degrees, since the dry air in passing through the main from the refrigerator to the blowing-engines was increased in temperature to 35 degrees F. From a careful indicating of the engines it had been found that there was 8.2 per cent. less weight of air entering the furnace than had been used prior to the introduction of the dry-blast. Next, with regard to the moisture contained in the atmosphere. It had been assumed to be a matter of common knowledge that the moisture in the air was contained as aqueous vapor, and when passed through the stoves became superheated steam, and whether the moisture entering the furnace as superheated steam with the blast, or as water from a leaking tuyere which was at once converted into superheated steam, it was nevertheless a form of water, and that it must be dissociated by the fuel and heat absorbed; but inasmuch as there was superheated steam to deal with at one end of the process, and frost or snow at the other end, it had been deemed preferable, as stated in the beginning of the paper, to represent the moisture contained in the atmosphere as grains of water per cubic foot of air.

In the paper presented to the Institute he had simply given the results obtained by the use of dry air, and without any intention of claiming that the economy in fuel was represented entirely by the weight of moisture eliminated. It would re-

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quire a very simple calculation to show that it was not. The calculations covering a period of operation showed that the saving, directly due to the removal of moisture, represented 24 per cent. of the total fuel saved. But a far greater economy was effected in the use of the dry-blast, and was represented by the reduction in temperature of the escaping gases and the greater efficiency in the reducing gases as shown by the increase in percentage of carbonic anhydride. These items alone would account for one-half of the saving in coke. In addition to the items already mentioned there was a material saving through the increased temperature of the hot-blast, a lowering of silicon in the metal, and in a smaller amount of slag to be melted, as less coke was required per ton of iron.

The moisture charged into the furnace through the raw material was many times greater than would enter the furnace from the atmosphere; but to get rid of it through simple evaporation by gases, which had already performed their useful work, was quite a different proposition from that of dissociating the quantity of moisture carried in from the atmosphere into the hearth of the furnace where temperature was such a vital element.

During the month of December there had been an excellent opportunity of noting the results obtained by the use of dry-blast, as Nos. 1 and 3 furnaces at the Isabella plant (No. 2 furnace having been out of blast) were making the same grade of iron. These furnaces were nearly the same size, No. 3 being a little larger. The same ore mixture and fuel were used in each furnace, and the conditions under which they were operated were identical. No. 1 furnace was blown with dry-blast containing 1 grain of moisture per cubic foot of air, while No. 3 furnace used natural air, which, being drawn from the engine-room, contained 2 to 4 grains of moisture. Although this period represented the best conditions as to dryness of the air in that climate, and furnaces under ordinary conditions increased decidedly in output above the average, yet the results obtained from a small and uniform content of moisture in the dried air supplied to the blowing-engines, in contrast with the varying content in the atmosphere, were very striking. From December 1 to 22, inclusive, the results were as follows:

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	No. 1 Furnace. Dry-Blast.	No. 3 Furnace.
Average daily product, . . .	449 tons.	400 tons.
Average coke consumption, . .	1,858 lbs.	2,309 lbs.

As connections had been made to the dry-blast main for four blowing-engines, and as only three had been used on No. 1 furnace, it was decided to connect the fourth engine to the main, and supply No. 3 furnace with one-third dry-blast. This was done on December 23, and it was decided not to make any increase in the burden or to change the revolutions of the blowing-engines, in order to see what effect the use of approximately one-third dry air would have on the output of iron. The furnace immediately began to drive faster, and the average daily product from December 23 to 31, inclusive, was 461 tons, an increase of 61 tons per day.

The increased efficiency of the furnace could be directed, in the main, to either increased output or to economy of fuel, according to the location of the works and as the commercial problem suggested.

The drying of the blast found, to some extent, its equivalent in an increase of temperature in the hot-blast which was wholly available, as the removal of the moisture represented a calorific gain in the furnace hearth proportionate to the quantity of moisture removed; but the dry-blast provided uniformity, which the hot-blast did not provide.

ALEXANDRE POURCEL, Paris, France (communication to the Secretary, received January 30, 1905)[4]: It seems *a priori* highly probable, in fact almost certain, as Mr. Lodin has said in his communication to the Academy of Sciences,[5] that the economy of fuel and motive power per ton of pig-iron which might be realized in Europe from Mr. Gayley's invention would not offset the cost of the installation and maintenance of the machinery and apparatus necessary for the desiccation of the blast.

There are on the Continent, even in France, blast-furnaces which consume only 950 kg. of coke per metric ton of pig, with a charge yielding but 33 to 34 per cent. of iron (instead of 43.5 per cent., as at the Isabella furnace), and a fuel contain-

[4] Translated from *Bulletin de la Société d'Encouragement pour l'Industrie Nationale*, vol. cvii., No. 1, January 31, 1905, p. 29, of the section, La Revue de Metallurgie.

[5] *Genie Civil*, December 10, 1904.

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ing 15 per cent. of ash and 5 per cent. of water (instead of 11.5 per cent. of ash, like the coke used by Mr. Gayley, who does not give the percentage of water).

It appears almost certain that with a charge producing 43.5 per cent., these furnaces would reduce their consumption of coke per ton to 850 or even 820 kilograms.

It is understood that the main object at these furnaces is the production, for the open-hearth process, of a basic pig-iron, containing less than 1 per cent. of Si, with from 1.5 to 2 per cent. of Mn, from 0.05 to 0.06 of S, and 1.80 of phosphorus.

But the blast at these furnaces is heated to 700° C. and more (instead of from 375° to 400°); moreover, the internal size of the stack varies from 2.5 to 3 cu. m. per ton of daily product; while at the Isabella, with an internal size of 512 cu. m., Mr. Gayley secured from moist blast a daily product of 358 tons, which was raised to 447 tons with the dry-blast.

Mr. Gayley, we must remember, has been the promoter of rapid steps of progress. His communication on blast-furnace practice to the New York meeting in 1890 of the Iron and Steel Institute made a sensation. The type of blast-furnace which he recommended (that of the Edgar Thomson Works) had a height of 27 m. and a capacity of 550 cu. m., and, for a production of 350 tons in 24 hours of acid Bessemer pig, the consumption of coke per ton varied from 775 to 840 kg. per ton. The ore yielded 62 per cent. of iron, and the coke contained 10 per cent. of ash. The temperature of the blast varied from 600° to 625° C., and the volume was 730 cu. m. per minute.

With rich and highly reducible ores, like those of Lake Superior, especially the soft hematites of Michigan, which are almost identical with the old La Vena ore of Bilbao (now exhausted), there is no inconvenience in rapid running—quite the contrary. But this is not the case with the ores of the East and of Luxembourg, as experience has proved.

The production from these ores in a furnace of from 500 to 550 cu. m. capacity seldom exceeded 150 tons in 24 hours at the normal rate of running for "Thomas" pig-iron; and if the temperature of the blast at these furnaces is kept as regularly as possible, above 700° C., the pressure established in the blowing-engines rarely reaches 40 cm. of mercury.

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Under conditions of operation so widely different from those of the Isabella blast-furnaces, there might well be a loss of economy of motive power, instead of a gain more proportionally approaching that of 136 h.p., which Mr. Gayley says he has realized.

In any case, the fuel-economy realized at the Isabella may have many explanations. As Mr. H. Le Chatelier has very courteously said, "we cannot doubt the exactness of the facts stated by Mr. Gayley," and the scientific explanation which he gives has a very real value; nevertheless, the separation of aqueous vapor from the moist blast appears also to deserve to be taken into account.

Mr. Gayley did not push the production from 358 to 447 tons until regularity of working appeared to him well established by the uniform descent of the charge, the absence of almost periodical slips of more or less voluminous agglomerated masses of half-reduced material, separated from the charge, the diminution and the uniformity of the temperature of the tunnel-head gases, the confinement within normal limits of the variation of blast-temperature between two changes of apparatus, etc. His increase of production was obtained with a smaller quantity of blast per minute—960, instead of 1,130 cubic meters.

That his hot-blast apparatus was able to maintain the temperature at 465° instead of 375° C. is explained: (1) By the smaller weight of the blast; (2) By the decrease of aqueous vapor contained in the dried air, the specific heat of steam being more than double that of air, and proportionally affecting the process of raising the temperature in the stoves; (3) By the corresponding decrease in the proportion of aqueous vapor contained in the tunnel-head gases. These gases, though poorer in CO (20, in place of 22.3 per cent.), have, by reason of their smaller proportion of aqueous vapor, a higher temperature of combustion.

The heat developed at the tuyere-level would be increased in notable proportion, in order that the same quantity of coke per charge may be able to support the increase of ore and flux indicated by Mr. Gayley, without marked alteration in the composition of the iron obtained, apart from the very certain decrease of manganese. In fact, as Mr. H. Le Chatelier has

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convincingly demonstrated, the desulphurization of the metal as a result of the elimination of aqueous vapor, from the blast, permits a lower degree of basicity, and consequent higher fusibility, with lower specific heat (nearer 400 than 500 calories), of the slag; but the proportion of manganese incorporated in the metal is thus diminished. Nevertheless, this is not an objection in a Martin-Siemens basic pig, so long as it has the essential qualifications of low-silicon and sulphur. In Luxembourg and in Germany, such an iron is designated by the mark "M. O."; that is to say, without manganese.

It is almost evident, without further argument, that all this combination of circumstances should tend to increase very sensibly, through the use of dry air, the quantity of available calories in the crucible. For the desiccation of the blast does away with the loss of heat at the tuyeres, due to the dissociation of the aqueous vapor of the blast, and also with the subsequent rise in temperature resulting from its re-constitution ($CO_2 + H_2 = H_2O + CO$) in the neighboring zone of the bosh, which renders viscous the fusible matter in the charge, forming an agglomerate which adheres to the walls. The periodical fall into the crucible of portions of these pasty half-reduced masses, when the accumulations which they form on the walls become too heavy to adhere, chills the zone below, and, consequently, debases more or less the quality of the pig-iron.

In other words, the temperature is lowered in the zone where its rise would have the pernicious effect of rendering the charge soft, pasty and tight, while, on the other hand, the zone in which carbonic oxide is liable to be decomposed into $C + CO_2$, so as to cement the ore, is extended. The diffusion of C in the ore, by the decomposition of CO, begins below 200° and reaches its greatest activity at 450°, terminating at 800° C. This kind of cementation is above all the most important coefficient economy of fuel in the reduction of the ore. The more we prolong this reaction by enlarging the zone in which it is produced, the more we can diminish the proportion of CO in the tunnel-head gases at the mouth of the furnace, and, consequently, lower the proportion of coke consumed per ton of iron.

It is then not without reason that we place the effects of the dissociation of aqueous vapor among the causes which might

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explain the economy shown at the Isabella furnace as a result of the use of dry air.

Mr. Gayley has noted also, as a consequence of the desiccation of the blast, a decrease in the amount of phosphorus in the iron produced, and a decrease in the loss of fine ore, carried out of the furnace by the gases, from 5 to 1 per cent.

The first fact may be explained only through the saving in coke, if the ash from it is high in phosphorus (which is not stated). The second fact should be due to the lowering of the gases at the mouth of the furnace, resulting from the acknowledged lowering of their temperature from 281° to 191° C.

It is always dangerous to make predictions; and, in the present case, one would run the risk of being a bad prophet in predicting that the great manufacturers of iron on our Continent will never find an immediate profit by putting in practice the idea of Mr. Gayley. The idea in itself is correct: Drying the air blown into a blast-furnace may assure regularity of running, and more or less economy of fuel; but it may remain to discover a method of realizing this end less complex, and, above all, less onerous than the one by the proposal of which Mr. Gayley has distinguished himself.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Influence of Carbon, Phosphorus, Manganese and Sulphur on the Tensile Strength of Open-Hearth Steel.

A discussion of the paper by Mr. Campbell which was read by title at the Lake Superior meeting, but first presented at the New York meeting of the Iron and Steel Institute (October, 1904) and here published under a mutual agreement between the Councils of the two Institutes.

MR. WILLIAM R. WEBSTER, Philadelphia, Pa., said: We are under great obligation to Mr. Campbell for his exhaustive investigation, and the results he has put before us to-day. They are the most complete series of tests that have ever been made, where all the conditions of rolling, etc., were kept uniform.

In discussing Mr. Campbell's first investigations, I claimed first, that the method of least squares, then used, would of necessity only give the average value of any element, and that if the effect of any element was greater in the presence of high carbon than with low carbon this would not be shown by the method of least squares. Secondly, that the effect of manganese should be considered on acid steel as well as on basic steel, instead of ignoring all the effect of that element when it was below 0.60 per cent., as was then recommended by Mr. Campbell. I am pleased to note that the results of his present investigation confirm my views on these points.

In presenting the results of his first investigation, Mr. Campbell referred to my investigation of 1893–94, and took exception to the value of phosphorus varying (depending on the amount of carbon present) from 800 to 1,500 lb. for each 0.01 per cent. He claimed that one might just as well consider the effect of phosphorus as being constant, and of carbon varying. I refer to this, as he now has introduced very similar conditions in his present tables, in that he gives variable values for manganese (depending on the amount of carbon present) for both acid and basic steels. Can we now say, with more certainty than formerly, which element should be considered as having a constant effect?

There is one other point that I took up in my former discus-

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sions, it is the matter of grouping the tests together, and getting an average of each element in the steels of the group—and from these average results working out the values. I worked with the individual tests and analyses, and consider them much better. I trust Mr. Campbell will give us the full results for each individual test, in order that others may be able to work out the values for each element by their methods. This, no doubt, would result in bringing into line the views of the different investigators on this important subject. For instance, would not the individual tests and analyses assist in deciding whether phosphorus should vary from one to one and seven-eighths times the effect of carbon, as I have it in my paper before this Institute in 1894? Or should the effect of phosphorus be constant and that of manganese vary in accordance with the amount of carbon present, as Mr. Campbell has it in his present paper? But, of course, it is much better to keep the effect of carbon per unit constant under all conditions, if possible, as that is the element we depend on the most.

Mr. Campbell's present investigation will greatly assist in harmonizing the work of others, as they have all worked with the method of "successive approximations," that is, "cutting and trying," and it will now be a very simple matter to compare results. In my own case I have no disposition to insist on the former value given for each element, which, though based upon numerous careful observations and proved by many subsequent tests to be approximately reliable in practice, are still open for correction, and will be unhesitatingly withdrawn whenever any other shall be shown to fit the observed facts more closely.

The practical value of the estimated ultimate strengths is now recognized by the steel manufacturers, and they use them in their everyday work. The color carbon determinations are used on account of the quickness of the method. I have not had time to study Mr. Campbell's paper as it deserves, but I have attempted to put his results in convenient form for comparison with those of Mr. Cunningham and my own. A table prepared in 1902 gives the values for each 0.01 per cent. phosphorus up to 0.08 per cent., in connection with each five points of carbon from 0.06 per cent to 0.60 per cent., and each five points of manganese from 0.20 per cent. to 0.60 per cent.

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Using Mr. Campbell's new values for carbon, phosphorus and manganese for acid and basic steels, and the carbon by both color and combustion, a new table was prepared, and a comparison of the results shows that, notwithstanding the differences of the values of each element used by the different investigators, the estimated ultimate strength by the different methods agree much more closely than we would expect. It would therefore seem to me that this matter will yet be very much simplified. For instance, have we enough data before us to say that the same amount of carbon in acid steel has from 30 to 40 per cent. greater effect than in basic steel? In the present case this greater value for carbon in acid steel is compensated for by giving the manganese a much greater value in basic steel, and also by using a higher value for pure iron in basic steel than in acid steel.

The factor "R," given in all equations as a variable to allow for heat-treatment, is of the most importance, as it has considerable effect on the physical properties of the steel. Formerly it was much neglected, but since the mills have been grading the steel by the estimated ultimate strengths, the heat-treatment in rolling has been much more closely watched. In 1894 I made a strong plea for an investigation to be made on the heat-treatment of steel in connection with the work of rolling and forging. A great deal has been done in this line since then, but there is much still to be done.

I agree with Mr. Campbell in that it is not necessary to take the microstructure of the steel into consideration from the standpoint of his investigation. Yet anything that will in any way assist in controlling the heat-treatment of the steel should be looked into. In the ordinary microscopical work they have not tied up the fractures of nicked and broken pieces of steel, as we know it with the microstructure of the same steel. This step from the old to the new has long been needed, and I now desire to call attention to a method of slight etching and low magnification with a hand-glass that is very promising. It looks as though by this method we will be able to tie up the fractures of steel of, say, 0.50 carbon and under. These pieces are from the same bar of 0.35 carbon acid open-hearth steel. One piece was overheated and shows the large coarse structure; the other piece was overheated and then annealed; it shows a

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much finer structure. A test of this kind would be useful in the case of large driving-axles, as a small spot could be polished and etched; the glass would show if the steel had been finished at too high a temperature in forging; that is, if it had too large a grain and in a dangerous condition. This method of investigation is offered at this time as a suggestion, in the hopes that others will take it up and improve on it.

Dr. Charles B. Dudley (Altoona, Pa.) said: I would like to say, with regard to Mr. Campbell's paper, that it has been known for years, I think, that there is a relation between the chemistry and the physical properties of a piece of steel. In the course of our work in connection with the Pennsylvania Railroad, we not infrequently have to examine broken parts. We do so by making both physical test and chemical analysis, and for a long time we have been impressed with a close relation between the chemical analysis and the physical properties. We ourselves, however, have never made any attempt to put this relation into figures. We can but admire the enormous amount of work that has been put upon this subject by Mr. Webster and Mr. Cunningham, and last and perhaps best of all by Mr. Campbell.

There is one phase of the question which Mr. Campbell does not claim for his work, and which we find very useful, namely, at times it is not possible to get enough of a broken part to make a tensile test. In that case we make an analysis, and interpret that analysis into a tensile test. We are constantly doing this, and are willing to say that the information so obtained is of great value. Furthermore, as is well known, most of the specifications of the Pennsylvania Railroad are both physical and chemical, and in making those specifications it is necessary that the chemistry and physical properties should agree, and here again is another place where we use the figures obtained by Mr. Campbell. To my mind his work is worthy of all praise.

Mr. Robert W. Hunt (Chicago) said: It gives me great pleasure to add my tribute in praise of Mr. Campbell's work. In the old days I do not know that I should have been so glad to have had the definite formula for steel presented, with the definite order as to exactly how the process of manufacture should proceed, the chemistry prescribed, and certain physical

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results demanded. But now that I am on the other side of the House, I think it well that these prescriptions should be made. The great point is that they should be intelligently given out; that is, based upon such information as to-day intelligent research presents to you. We all, who have been connected with the steel industry for many years, know that for a long time steel was regarded as a sort of unknown problem, and a hopeless one, so that its idiosyncrasies should be accepted with regret, but still accepted. But that time is passing away. Now we can pretty nearly always tell why those peculiarities occur. I have no doubt as the investigations proceed the time will happily come when we can always tell. Dr. Dudley has done more than any other expert in this country to draw correct deductions from the points that he mentioned to-day; that is, from the failures. In some of the discussions of the American Society of Civil Engineers the value of the scrap-pile has been enlarged upon; notably by one of its past-presidents; and that scrap-pile is a lexicon from which all manufacturers of steel can derive valuable knowledge.

Dr. J. A. Mathews (Syracuse) said that his experience had been entirely in other grades of steel than those which Mr. Campbell had brought forward, so he did not think he could add anything of value. The relation between the manganese and the sulphur might be considered specially, inasmuch as it was generally considered that the manganese and the sulphur combined chemically so far as there was an excess of manganese, and then the free manganese uncombined might exert an effect independent of the total manganese. That was merely one phase that occurred to him as being worthy of looking into. He was sorry that his experience had not taken him into this field. He appreciated very greatly what Mr. Campbell had done, and thought it would be of immense value to the industry upon the structure of that grade of steel which is so important.

Mr. Campbell, in reply, said: Mr. Webster raised the question whether the lower value for carbon in basic steel was not due to the higher value of the base. A little calculation would show that this criticism applied only to a limited range of low carbon steels. The value of carbon in acid steel was 1,000 and for basic steel 770 pounds. In a steel of 0.30 per cent. carbon the effect of carbon in the one case would be 30,000 lb. and in

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the second 23,100, a difference of nearly 7,000 lb., while the base for basic steel is only 1,500 lb. higher than for acid metal. It is quite evident that no change in the value of the base can make up for an actual difference in the effect of carbon. A base can arbitrarily be assumed, and a value found for carbon for low steels, or another can be assumed and another value found for high steels, but if a formula is to fit both high and low steels, the base cannot be assumed, but must be worked out from the records. It was also suggested that a list of the individual heats should be printed, but it seemed unnecessary to burden the pages of the *Journal* with the chemical and physical records of 1,800 heats. The full record, however, would be given to Mr. Webster for his investigations.

A cordial vote of thanks to Mr. Campbell was passed unanimously.

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Improved Method of Slag-Treatment at Argo.

BY HAROLD V. PEARCE, DENVER, COLO.

(Washington Meeting, May, 1905.)

THE plant of the Boston and Colorado Smelting Company, at Argo, Colo., has not received special notice in technical or scientific publications for some time past. Dr. Peters[1] described the development of the reverberatory furnace used at Argo, but he dealt almost exclusively with the increase in size and the corresponding gain in capacity. Since the time of his description the changed conditions have led to a modification of the former smelting practice, and this modification forms the basis of the present paper. No claim is made for personal originality in connection with these improvements, in fact, I expressly disclaim any such intention, since nearly every successful step in the development of the process now used at Argo has been suggested by Dr. Richard Pearce at various times during the past 12 years, though perhaps in a different manner of execution. These changes would undoubtedly have been made before, were it not that many problems of greater metallurgical importance and momentarily of more urgent need thus had to be solved. At the time to which I refer, economic conditions demanded mainly an increase of capacity. Moreover, the determination and placing of exact losses in smelting were not so imperative as they now are.

About three years ago, the time of my appointment to the management of the works of the Boston and Colorado Smelting Co., the great decline in the price of copper occurred; the Argo plant, in common with all others engaged in the copper-industry, lost heavily in the market-value of the stocks on hand of ore and furnace-products. Added to this was the unfortunate circumstance that the copper-supply of the Argo plant from general sources began to dwindle; in fact, it almost entirely ceased. About this time the smelting-plants in the west-

[1] Modern Copper Smelting, 7th ed. (1895).

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ern and northwestern States, on which the Argo plant for years had relied for a large proportion of its supply of copper in the form of matte, added converters and Bessemerized their own copper-matte, thereby effecting a considerable saving in transportation- and smelting-charges. This local treatment of copper-mattes doubtless resulted from the development of the railroads, which simplified the question of fuel-supply and freight-rates, and, in justice to the former work at Argo, it should be said that there was *no* expression of dissatisfaction from those smelting concerns with whom the company had been dealing for many years on mutually satisfactory terms.

In this way, what had always been in latter years a fairly lucrative portion of the business of the Argo plant was practically lost. But to what proportion an abundant copper-supply entered into the economical necessities of the operations was neither realized nor accurately disclosed until this supply discontinued. A new system was then adopted in order to determine the exact losses or gains in gold, silver and copper; and these losses and gains were calculated for a number of years preceding, which showed that the only way to meet the altered conditions (since the smelting-charges could not be increased) was either to reduce the said losses in gold, silver and copper, or to make an increased gain where there had been a gain before; also to curtail operating-costs as far as possible.

The new situation required the adoption of radical changes. Operating-expenses were reduced more than ever before, but this economy alone could not counterbalance the greater deficiencies. New means had to be adopted to reduce losses in gold, silver and copper, which under more flourishing circumstances might have been regarded as unavoidable, but which had become a matter of vital importance and concern.

Following the research work which was directed toward the placing and the reduction of smelting-losses, there came a readjustment of furnace-work which has resulted in very important benefits and economies. It is not the purpose of this paper to publish comparative assays of slags before and after these particular changes in treatment went into effect, but to present an adequate idea of the savings effected thereby.

Although the lessons of the past have justified many improvements, every one knows the obstacles and difficulties which

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beset the manager who ventures to remodel a long-established smelting-plant such as the one at Argo.

About eighteen months ago experiments were started having in view the cleansing of the ore-furnace slags. Owing to the large capacity of the ore-furnaces and the high degree of concentration effected in them, a very large quantity of slag was produced; and the saving of even a small portion of the values contained therein would amount to a very substantial aggregate in a year's time. These experiments were made in large crucibles in the refinery under the direction of my assistant, Mr. F. C. Knight, the object being to discover the effect of tranquil settling under various conditions of temperature and time; also the effect of adding different quantities of a matte-forming material, such as iron pyrites, to the charge of slag.

Formerly, the slag was divided into "clean" and "foul," and, before loading into railroad-cars, both classes were roughly broken, picked over by hand, and any lumps showing signs of carrying prills of matte were thrown back to be resmelted as foul slag. The remainder of the slag, which constitutes, of course, the larger portion of the skim, was loaded on cars and sent away as clean slag. The point to be observed in this practice, however, is that, although a large portion of escaping values was recovered in the foul slag, it was necessary to resmelt a considerable quantity in order to secure it. On the other hand, the so-called clean slag undoubtedly carried with it a certain quantity of foul slag. The assay of the sample of clean slag did not give a fair indication of the quantity of foul slag contained in a car-load, since the whole car-load of solidified slag was never crushed; but only a small ladle-sample was taken while it was molten. The sorting of slag was done in the open-air under all conditions of weather, and consequently it was impossible to make a perfect separation.

The object of these experiments was not so much to find out how the clean slag could be made cleaner, but how to treat the mixture of foul and clean slags as produced in the ore-furnace during the process of skimming the charge. The general results of the experiments were that, by the addition of a certain quantity of clean pyrite to the molten mixture of foul and clean slag, subsequently allowing the mixture to remain quiescent at a fairly high temperature for a short time, a small quantity of

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low-grade matte was precipitated to the bottom of the crucible and the superimposed slag was no longer a mixture of foul and clean slags, but a very clean slag indeed. No prills of metal could be discovered, and all the values in the foul slag that had been originally in the mixture were concentrated in the small quantity of matte at the bottom. The treatment of this matte on a large scale is a very simple matter, as compared with the former practice of handling and re-smelting all the foul slag.

These results, indicating, in part, a solution of the problem, it was decided to build a small auxiliary reverberatory furnace for each of the large ore-furnaces, the idea being to skim all the slag from the latter into the smaller furnace, where the cleansing could be effected in a manner similar to that of the crucible experiments.

It had been the former custom at Argo to skim the slag from the large ore-smelting furnaces through two doors on either side, four skimmers being employed in this work. The first auxiliary slag-furnace, as it is called, was built near the end of the ore-furnace, so that the slag skimmed from both sides could most conveniently be conveyed to it by means of iron spouts. The large quantity of slag, which became chilled in the spout leading from the more remote side, however, was a great drawback, and it was then decided to skim from one side only, the four skimmers taking turns at the rabbles. This arrangement seemed to work very well, and the later slag-furnaces were built by the side of the big ones, as shown in Figs. 1, 2 and 3.

It was necessary to make many tests in order to determine the proper quantity of sulphide required to clean the slag on a large scale and under the different conditions of furnace-temperature. Many samples of clean slag were carefully taken and treated with various quantities of sulphide, subjecting the result to varying periods of settling. The samples were sent to an independent assayer who was instructed to show no favor in the determination of slag-losses. In this way the most profitable conditions were ascertained, and, at the same time, assays were obtained that were absolutely reliable, since they were made by an impartial assayer who eliminated the personal equation that might have been induced by the desire for low results.

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FIG. 1.—TAPPING SLAG FROM ORE-FURNACE TO THE AUXILIARY SLAG-FURNACE, WHICH IS BUILT AT THE SIDE, AS SHOWN IN FIG. —.

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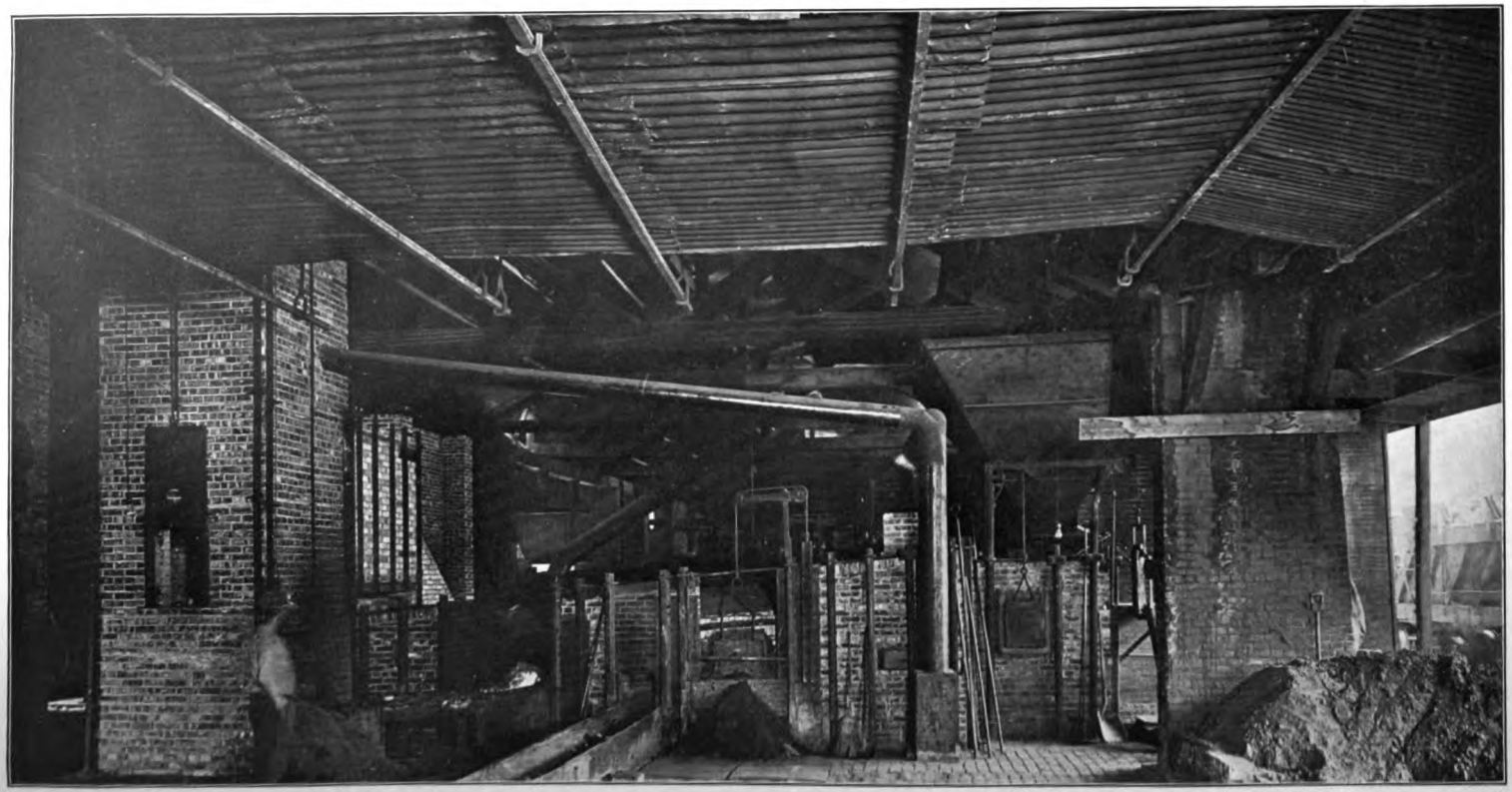

FIG. 2.—AUXILIARY SLAG-CLEANING FURNACE, READY FOR TAPPING THE CLEANED SLAG, THE TAP-HOLE FOR MATTE IS AT THE RIGHT AND THE CONICAL PILE BELOW THE SIDE DOOR IS A CHARGE OF SULPHIDE MATERIAL.

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The time of "cleaning" a charge of slag in an auxiliary furnace is limited to the time occupied by the ore-furnaces in smelting a charge of ore. The slag-furnace must always be ready to treat the skim from the ore-furnace, which latter should not be kept waiting. About three hours and a half are usually required for the smelting of a charge of about 12 tons of ore.

The method of cleaning the slag is as follows: The large ore-furnace being ready for skimming, the spout is lowered in position which connects it to the small auxiliary slag-furnace. The slag-furnace fireman then stands ready to direct the flow of slag onto the hearth of the latter. The two side skimming-doors of the ore-furnace are removed, and four skimmers take turns at the skimming-rabbles, as before stated. After all the slag has been removed from the ore-furnace, that portion which became chilled in the spout is at once pushed into the slag-furnace and the door closed. The fireman then stokes the fire and maintains a moderate heat for about an hour, which suffices to melt the "spouts," as the chilled slag in the spout is called. The side door is then raised and 1,000 lb. of sulphide, usually in the form of "tailings," are sprinkled evenly over the surface of the liquid slag. More heat is then applied and the settling and cleansing period begins, continuing for two hours or more, according to the requirements of the ore-furnace. In Fig. 3 it will be noted that the slag is tapped and not skimmed from the slag-furnace after it is cleaned, the hearth being built to slope rapidly toward the matte tap-hole, and the slag tap-hole being placed a short distance away at a slightly higher level. The level of the matte underneath is easily determined, and the slag is tapped almost down to this level. A "bay" or dam, consisting of clay and sand (see Fig. 3), is built up in the slag tap-hole of the slag-furnace, as the matte accumulates on the bottom, and the level of the slag rises in proportion. When ready to tap the slag, the temperature of the furnace is slightly raised and the slag-furnaceman tears down the dam gradually by means of an iron hook almost to the level of the underlying matte,—an action which allows the slag to flow out into sand molds conveniently placed to receive it. The slag tap-hole is then built up again with sand, and the furnace is ready to receive the next skim from the ore-furnace.

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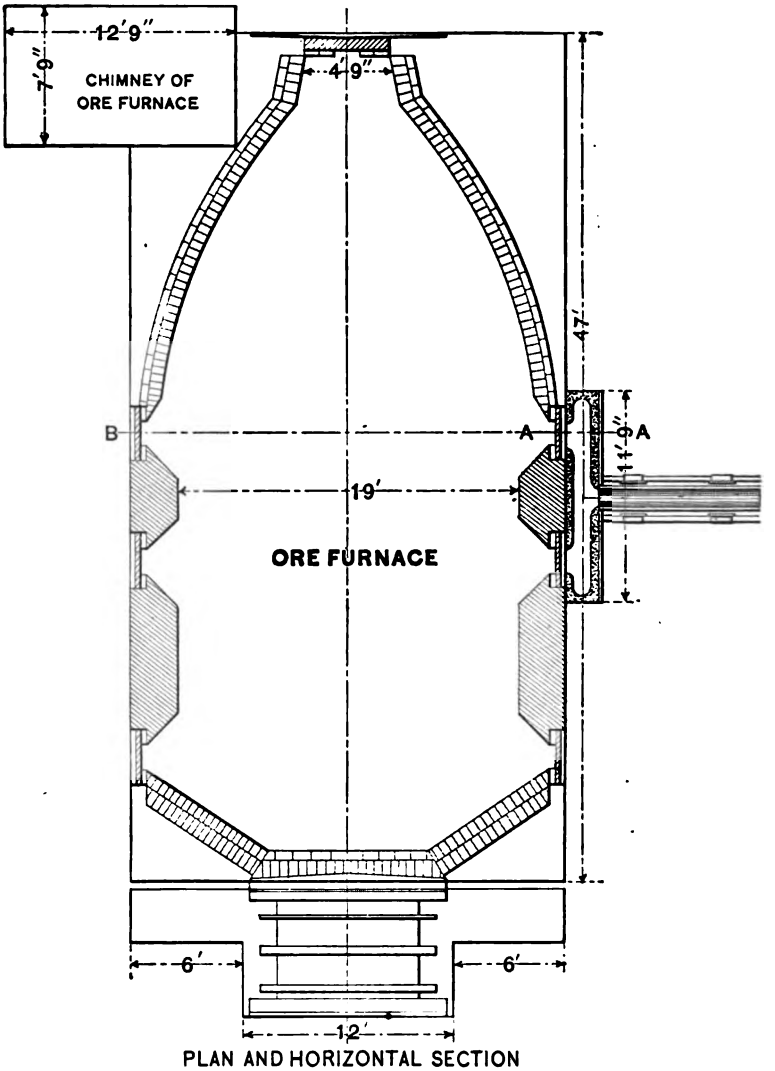
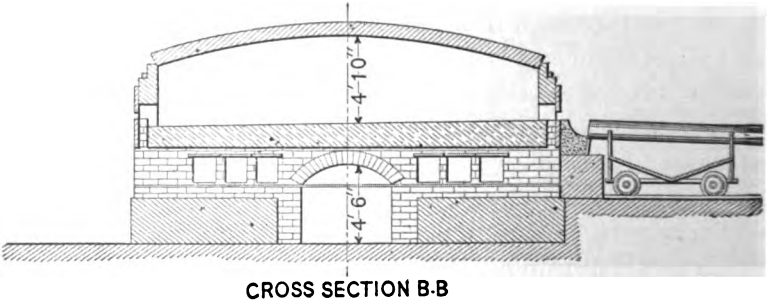


PLAN AND HORIZONTAL SECTION



CROSS SECTION B-B

FIG. 3.—PLAN AND SECTION OF THE ORE-

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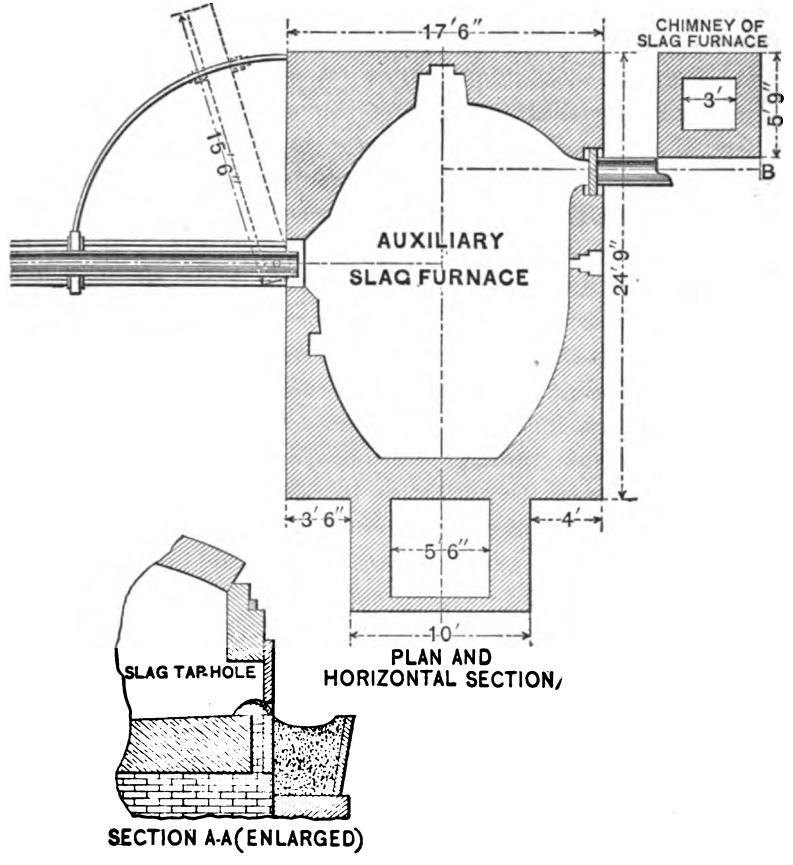

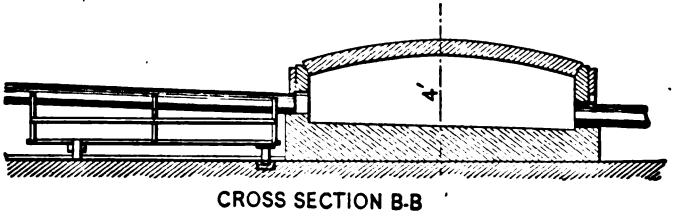

CROSS SECTION B-B

FURNACE AND THE SLAG-FURNACES AT ARGO.

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After treating about 20 skims of slag in this manner, the slag-furnace contains 4 or 5 tons of low-grade matte, which is then tapped through the matte tap-hole in the ordinary manner; but before letting out this matte, the small quantity of slag remaining on its surface (which is always left behind as a safeguard against the escape of small quantities of matte at the end of each out-flowing of slag) is skimmed through the front door of the furnace.

In describing the operating of the slag-furnace, it will be noticed that I have always used the terms "skimming" and "skims" in referring to the slag tapped from the ore-smelting furnaces, and in this connection a point arises which I judge is of great importance to those using reverberatory furnaces in lieu of, or in addition to, blast-furnaces.

It had long been a cherished hope that some day and somehow the laborious and somewhat painful process of skimming by the hand-manipulation of the venerable rabble might be dispensed with, but, for a long time, we were never able to summon up sufficient courage to try any of the several methods which have occasionally suggested themselves. The experience gained in operating the auxiliary slag-furnaces, however, furnished the necessary encouragement. The art of building up the bay or dam in the slag tap-hole of the small slag-furnace, and the control of the out-flow of slag therefrom, having been mastered, a fairly safe basis was thus afforded on which to attempt to regulate the flow of the slags from the large ore-smelting furnaces; moreover, the certain knowledge that any matte which might escape in the flow of slag from the ore-furnace would be caught by the auxiliary slag-furnace removed any anxiety concerning the result of the trial so far as losses were concerned. The trial made on one furnace was so eminently satisfactory that skimming, so far as the large ore-furnaces are concerned, has been discontinued.

The flowing of the slag takes place simultaneously through the two side-doors of the ore-furnace, and the bays or dams are built up precisely as in the slag tap-hole of the slag-furnace. Rabbles are used for stirring the charge and for what is called "trying" the furnace. It will be noticed in Fig. 2 that the side doors are suspended from the short arm of a lever, so that the skimmers (or rather "slag-tappers") themselves can raise

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the doors and regulate the opening by means of the slots which are arranged to engage the long end of the lever. This arrangement simplifies matters very much.

Practical smelting men will appreciate the full meaning of this method of tapping slag when it is noted that lately at three large ore-furnaces the smelting of the charges had become finished at the same time; in other words, the slags of the three furnaces were ready to be skimmed or tapped at once. Although this occurrence is unusual, if it had happened under the old skimming-system, especially in hot weather, general despair and pandemonium would have resulted. Under the new conditions, however, two slag-tappers handled the three furnaces with perfect ease, casually strolling from one furnace to another, and keeping an eye on the out-flow of slag.

Figs. 1 and 2 show the operation of slag-flowing. Formerly two head-skimmers and two helpers were employed on each shift, but now the three large furnaces are handled by two head-skimmers in each shift. The economy here is apparent.

It may be mentioned that, as is also the case with the slag-furnaces, a small quantity of slag remains on the matte in the ore-furnace after the slag has been tapped. When this matte is ready to be tapped, say, once a week, the slag is skimmed off in the old way.

By the method of tapping the slag, a saving of thousands of dollars a year is made in rabble-iron alone. There is, also, a saving of soap which was formerly used to make the rabbles slide easily back and forth over the skimming-bar. Another item is the saving of elbow-grease, which can be better imagined than described.

I am not aware that tapping the slag, as described above, has been done elsewhere; possibly there is nothing novel in it; but for the benefit of those engaged in reverberatory smelting, to whom the use of auxiliary slag-cleaning furnaces would be of no particular benefit, it will be of interest to know that the use of the tapping-method, if properly carried out, results in a much cleaner slag than under the old method of skimming. In the practice at Argo, there has been a considerable decrease in the tenor of the matte resulting from the cleaning of the slag in the slag-furnaces since the tapping method has been adopted.

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Repairing Partly Collapsed Cylindrical Furnaces.

BY JOHN P. COSGRO, ROSSLAND, B. C.

(Washington Meeting, May, 1905.)

THE increasing use of internal furnace-boilers for mining power-plants (doubtless due to the facility with which they may be installed by reason of their portability; the fact that they require no masonry setting, cast-iron fronts, buck-stays, etc., and practically no foundation; and the successful maintenance of their claim to equal and even superior economy, as compared with boilers of other types[1]) warrants this description of a method of remedying an injury which may occur, to such a furnace, even though it satisfies the requirements of the formula adopted by the U. S. Board of Supervising Inspectors of Steam Vessels.[2] I refer to the sagging and deformation which may result from overloading the boilers for a considerable period—say, several days. This is sometimes unavoidable under the exacting conditions of continuous day-and-night service, and (in spite of all rules and precautions, the occasional careless or unskillful work of attendants. Such a trouble is by no means unknown on shipboard, where this type of boiler is so largely employed.

In the case here described, two Morrison suspension-furnaces, of half-inch steel, 14 ft. long and 45 in. in mean diameter, with "dry-back" boilers, were necessarily operated somewhat beyond their rated normal capacity for several months, while a third similar unit was in process of installation, and during this period, though the boilers were kept perfectly clean, heat seems to have been generated slightly faster than it was absorbed by the water in the boilers. The corrugations of this

[1] *Engineering Magazine*, April, 1904, page 91.

[2] This formula is $P = \frac{15000}{D} \times T$, in which T is the thickness of furnace in inches; P, working-pressure in pounds per sq. in.; 15000, a constant; and D = mean diameter (least iside diameter, plus thickness of metal, plus 1.5 in.)

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type of furnace have the form of a catenary, with 8-in. span and a versed sine of 1.5 in., and, as a result of the abnormal strain above described, the furnaces had partly collapsed, or dropped a total distance of $1\frac{3}{16}$ in. As would be expected, the greatest deformation was just back of the bridge-wall; and since external pressure always tends to increase greatly the least departure from a true cylindrical cross-section, forcing it to become more and more elliptical, and finally to collapse and fall, it was advisable to correct immediately the change already effected, by restoring the original form of the furnaces.

The method proposed by the Government boiler-inspector, namely, to heat the metal, and press it up into place with a jack-screw, was impracticable, because, when half-inch steel shell is hot enough to be set with a jack-screw, it is altogether too hot for a man to work in.

The work was done successfully by means of a simple hydraulic press, designed for the purpose, of which Figs. 1 and 2 are cross-sections, Fig. 3 a right-line drawing prepared from a photograph, and Fig. 4 a sketch showing the general arrangement and method of operation.

A natural hydrostatic pressure of about 250 lb. per sq. in. was available from a tank on a hill several hundred feet above the boiler-house, through the pipe-line of the general fire- and water-service. Using this pressure direct, we obtained by means of the $10\frac{1}{8}$-in. lifting-cylinder, L, Fig. 1, a lifting-force of 10 tons at the pressure-head, B_1.

No data being obtainable as to the force necessary to press up the dropped crowns, it was seen that the required lifting-capacity must be largely a matter of judgment. At the time, it seemed to me unlikely that more than 20 or 30 tons would be needed; but, in view of other work then in contemplation, a capacity of 50 tons was provided; all of which was in fact required in re-shaping the furnaces.

A flexible conduit, a piece of new 1-in. air-hose taken from stock, was connected from the pipe-line to the pressure-head, as shown in Fig. 3, and gave satisfactory results, bearing 250 lb. without showing any signs of distress. It could not be expected, however, to stand the pressure necessary for a lifting-effort of 50 tons; hence an intensifier with a low-pressure plunger $4\frac{1}{8}$ in. in diameter, a high-pressure plunger $1\frac{3}{16}$ in. in

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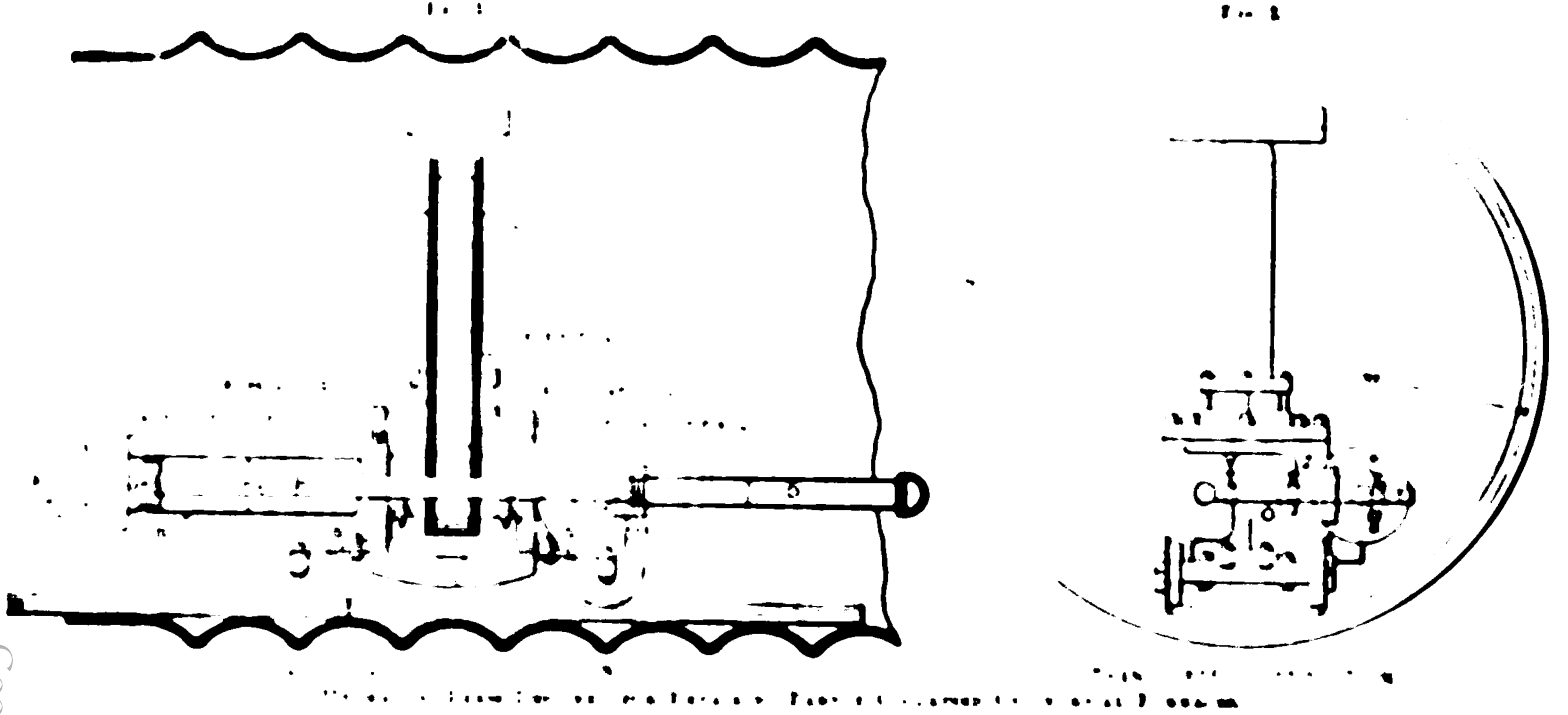


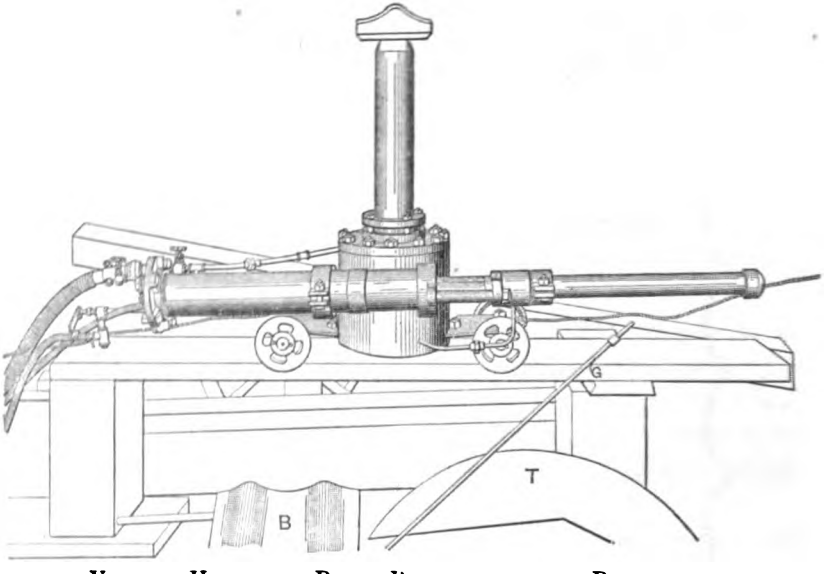
diameter, and a 24-in. stroke, was mounted on the press itself with its center of gravity over the longitudinal center of the press, and connected to the lifting side of *L* by the high-pressure pipe, *H*.

By calculation based upon the data given, it will be found that the delivery-pressure due to intensifying was

$$\frac{\text{Area of low-pressure cylinder}}{\text{Area of high-pressure cylinder}} \times 250 \text{ (the line-pressure)} =$$

1,300 lb. per sq. in., and that the displacement of the high-

FIG. 3.



VIEW OF HYDRAULIC PRESS, PREPARED FROM A PHOTOGRAPH.

pressure cylinder per stroke was the area of that cylinder × 24 = 61.9 cub. inch. Hence the intensifier would lift the ram a distance of 0.75 in., and exert a force of 52 tons, neglecting friction, weight of ram, etc.

The idle part of the stroke being effected and all slack taken up by the line-pressure, practically all of the high-pressure ram-movement could be utilized for pressing up the crown.

The mode and sequence of operations was as follows: The part of the furnace where operations were to begin, and at the places shown to be out of round by template, *T*, Fig. 3, were heated to low-red by a wood-fire, contained in a sheet-iron pan

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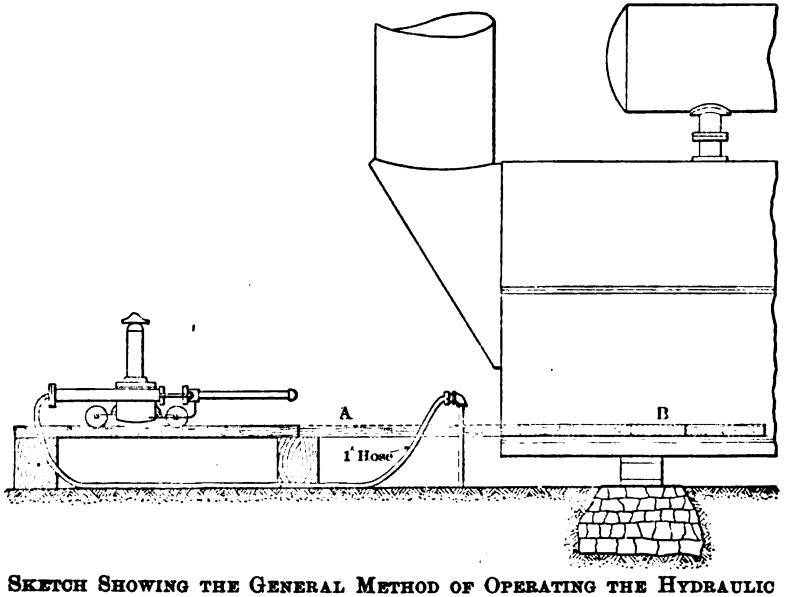
or basket, fitted with long handles to facilitate its easy handling. Directly under the spot to be pressed up was placed plate *B* (Fig. 3), a casting roughly fitting the corrugations of the furnace. This plate is 12 in. wide and fitted the tread of the truck-wheels on the press.

A short piece (say, 2 ft.) of plank, placed beyond and abutting against the plate, formed an extension-track; and longer pieces similarly laid to connect the plate with the bench, upon which the press is withdrawn and kept out of the way during

FIG. 4.



SKETCH SHOWING THE GENERAL METHOD OF OPERATING THE HYDRAULIC PRESS.

the heating, made a good track upon which the press could be readily moved in and out of the furnace (see Fig. 4). Plank *A* was taken out of the way when not in use.

When the ram was in its lowest position, as shown in Figs. 1 and 2, the plate, *B*, which was set in a ball-and-socket seat, and held in place by a loosely-fitting dowel-pin, had 2 in. clearance from the top of the furnace.

When a long low heat had been secured the fire-basket was taken out, and the plank, *A*, put in the furnace. The press was then put into place with the cylinder over the plate, *B*, and

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pressure was admitted through the 0.25-in. globe-valve, *Y*, from the pressure-header, *O*, to the lower end of *L*. This raises the ram and block, *B*, up against the furnace, and springs, *L*, down half an inch, when the rounded bottom comes in contact with *B*, from which it had previously been held apart by the spring-plates, *S S*.

In filling the lower part of *L*, the water also fills the high-pressure cylinder, *D*, through *H*, and acting on plunger, *P*, has sufficient force to drive it back to the position shown in Fig. 1.

When the pressure-block is hard up in place, water is admitted from *O* to low-pressure cylinder, *R*, through valve, *W*, which advances the plunger, *P*, in *D*, forcing the water back through *H* to *L*; escape through the admission-pipe being prevented by a check-valve placed between the admission-valve and cylinder.

The pressure in *L* is now 1,300 lb., or a total lifting-effect of 52 tons on a $10\frac{1}{2}$-in. piston.

As soon as the plunger, *P*, reaches the end of its stroke, valve *W* is closed and valve *X* is opened, when the line-water coming into *L*, through admission-pipe and again filling *D*, forces *P* back to the first position ready for another stroke. This action may be repeated indefinitely, and at each heat two corrugations or about 16 in. are treated.

By testing the curvature with template, *T*, tilting the press slightly from the vertical when necessary, and by comparing the vertical and the horizontal diameter with the caliper-gauge, the curvature of furnace may be brought up to a practically true circle.

The variation certainly need not exceed $\frac{1}{16}$ of an inch.

Details of Construction.

Low first cost, being necessary in making this experimental machine, stock on hand was utilized wherever possible. In fact, only the main cylinder, the head and plates, *B* (Fig. 3), and, B_1 (Fig. 1), together with the two caps for the intensifier, were purchased,—in all, about 300 lb. of rough castings.

The ram was made of a piece of 4-in. double extra heavy pipe—a size sometimes used in the mine for column-bars—turned to 4.25 in. in diameter. Cylinder, *D*, is a piece of 2-in. standard brass pipe specially threaded with 15 threads to the

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inch at each end. Cylinder, *R*, is simply a piece of 4-in. standard iron pipe, bored with one cut to 4⅛ in. fitted with a piston from an old 6 by 4 by 6 in. duplex pump screwed on to a piece of 1⅛ in. diamond-drill tubing, turned down to 1 1/16 in. and plugged, forming the plunger, *P*. Standard 0.25-in. globe and angle-valves were used for controlling admission and exhaust, and 0.25-in. standard iron pipe for all cylinder connections, except for *H*, which is ⅛ in. standard. I am aware that it is pressing the limit somewhat to put 1,300 lb. pressure on 0.25 standard pipe, but, as it stood up to the work, it shows what may be done under stress of circumstances. Single-cup leather packing of the usual form was used on the lifting- and intensifier-plungers. The cost of the press, figuring material and labor, at cost, was $67.

All the work of building a press of this kind may be readily handled in any mine machine-shop equipped with a drill-press and a lathe capable of swinging 18 in. or more. The patterns were made by a mine-carpenter having no knowledge of pattern-making.

The entire cost of repairing the two furnaces, including dismantling and refitting, cost of press, but not the superintendence, was $187. The time taken for the work was eight days.

This cost was not considered excessive, in view of the fact that a pair of new furnaces would have cost $700, each, f. o. b. Rossland, to say nothing of the expense involved in changing the furnaces, and remembering that, unless replaced or repaired, the old ones were not considered safe at the steam-pressure necessary for the economical operation of the engines.

The cost is much less, and the operation more satisfactory than that of a hydraulic jack to do the same work.

Remarks.

As a hydrostatic head of any considerable force is frequently not available, a small boiler-feed or tank-pump might be utilized to furnish the necessary pressure, or by rigging up a temporary accumulator, a hand force-pump could be used.

A somewhat lower water-pressure could be employed by increasing the size of the main cylinder, but, if this be carried to extreme, it would sacrifice mobility and ease of operation, two

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very essential features when a hot furnace is ready to be repaired.

An oil-burner could be used to excellent advantage to take the heats if the price of fuel-oil permits.

In order to insure against oversetting (an advisable precaution if inexperienced help is to be employed), a hole may be drilled in the main-cylinder wall at such a height that it will be uncovered and release the pressure when the piston has risen a proper distance. The intensifier is clamped to the main cylinder by two straps of 2 by $\frac{5}{8}$-in. iron bolted to pads, cast on the side of the cylinder for the purpose.

Furnaces should be tested with template, *T*, in order to determine at what points the collapse has started and the heat should then be taken at the points indicated. In every case the action of the press will be in a nearly vertical direction, the heat being taken at the place of deformation allowing correction at these points.

After having been in service for 14 months from the time the above repairs were made, the furnaces were calipered and found practically perfect.

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Fuel and Mineral Briquetting.

A Discussion of the Paper by Robert Schorr, read at the Atlantic City meeting, February, 1904.

(Washington Meeting, May, 1905.)

E. T. DUMBLE, Houston, Texas (communication to the Secretary*):—In addition to the list of publications mentioned by Mr. Schorr and those by Prof. Hofman, I call attention to the following references:—

Studi sulle Lignite, by Capacci. Turin (1890). (An excellent work.)

Report of Brown Coal Industry in Germany. J. Cosmo Newberry. Department of Mines, Victoria (1892).

Second Report of the Royal Commission on Coal Supply. Vol. II. London (1904).

Peat Fuel: Its Manufacture and Use. Report of the Bureau of Mines of Ontario, 1903.

Mr. Schorr says: "Brown coal and peat are briquetted as a rule without binder," but this statement is entirely true only so far as raw peat is concerned, and it is but partly true of brown coal.

With regard to raw peat, briquetting without bond is carried on successfully both in Europe and America, and the development of the industry in Canada is fully described in the Bureau of Mines report for 1903, mentioned above. While there is still room for improvement in the efficiency of some of the arrangements for drying the peat previous to briquetting, a very satisfactory fuel is made at a price which enables it to compete with coal.

During the late strike in the anthracite regions of Pennsylvania, the manufacture of peat-briquettes received a considerable impetus, and possibly the attention directed to it on this account may result in a considerable increase in the output. In Canada, the peat when it is first excavated from the ground

* Recived Aug. 27, 1904.

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contains more than 80 per cent. of moisture, which quantity is reduced, first by drying in air and then in artificial dryers until in the finished briquette there remains only 15 or 16 per cent. of water. The briquettes are cylindrical in shape, 2 in. long and 2 in. in diameter, contain from 1.55 to 10 per cent. of ash and is said to have a calorific value two-thirds that of anthracite coal. The estimated cost of excavating, drying and pressing peat is $1.01 per short ton, while the gross cost, including land, depreciation, interest and royalty (25c.) amounts to $1.80 per ton.

The earthy brown coal of the Rhine provinces and in the vicinity of Halle am Saale is the only variety of brown coal, so far as I know, which can be briquetted without bond. This variety of brown coal is very different from common brown coal and its derivatives, pitch-coal and glance-coal, and it is even claimed by some that earthy brown coal is not a true coal at all. It frequently contains retinite, amber and the pyropissite or "schweelkohle" which is the basis of the paraffine industry of Germany. Schweelkohle does not occur in other brown coals. In structure it is usually friable and earthy, although at times it is more compact and resembles common brown coal. In color it varies from yellow to brown or brownish black. Earthy brown coal when first it is mined contains from 30 to 60 per cent. of moisture, which is considerably reduced by drying in the process of briquetting, but the finished briquettes still carry from 14 to 18 per cent. of water. Consequently, after earthy brown coal has been briquetted, it is practically of the same composition as the common brown coal of Bohemia and Texas taken direct from the mine, but it is firm and clean, it will not slack on exposure to the air and it burns with little smoke, and, for these reasons, it is a very desirable fuel.

In the briquetting-works of Germany, of which there are 286, more than 44,000,000 tons of earthy brown coal are annually consumed.

So far as I know there is no earthy brown coal in the United States.

Common brown coal, pitch-coal and glance-coal, which constitute by far the greater bulk of all the brown-coal deposits, cannot be briquetted without bond under ordinary conditions;

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and numerous failures are on record of attempts to make serviceable briquettes of them even with the best of binders.

In connection with briquetting brown coal with a bond, Mr. Schorr refers to the lignite-briquetting plant built at Rockdale, Tex.

Under my direction the brown coal from Rockdale was experimented with in Europe in 1892, and a good firm fuel made from it by briquetting with a binder of hard pitch without other admixture. Therefore, there is every reason to expect that satisfactory briquetting can also be done in the United States by proper treatment. Later, the plant at Rockdale was taken up by San Antonio people, and a firm of mechanical engineers, of considerable experience in coal, was employed to investigate the subject of briquetting and to construct a plant. In 1896, after an examination of the various machines and appliances used in Europe, a plant was designed and erected which, it was believed, would meet the requirements; but so far as I can learn it never passed beyond the experimental stage. The first trouble encountered was to get a sufficient quantity of pitch at a suitable price; but this product was finally obtained from the iron blast-furnaces in Scotland. Then the quantity of moisture in the lignite could not be reduced to the exact point necessary for a good cohesion. In October, 1896, the plant burned down, but it was later rebuilt.

My information is that a fuel could not be made of sufficient firmness to be satisfactory. Whether this was the fault of the press or of the treatment I am not prepared to say, but the work was considered a failure. Such a failure as this, backed as it was by considerable money, has made capital wary of a second investment in this attractive field. Nevertheless, I am convinced that sooner or later the process of briquetting Texas brown coals with a binder of pitch or asphaltum will be successfully operated and that it will provide a satisfactory fuel for general use.

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The Commercial Wet Lead-Assay.

A Discussion of the Paper by Mr. H. A. Guess, read at the Atlantic City meeting, February, 1904.

MR. JOSEPH P. GAZZAM, Germiston, Transvaal, So. Africa (communication to the Secretary*):—About fourteen years ago, in southeastern Missouri, I used a method for the wet determination of lead which closely resembles the modified sulphate-chromate method described by Mr. Guess. The principal difference, however, is, that after dissolving the lead chromate in hydrochloric acid, the chromic acid is determined by means of a standard solution of ferrous sulphate. The method, which I believe is still in use in southeastern Missouri, was found to be convenient and accurate for the determination of lead in the ores and various products of the mills and furnaces in that district, but it was never used for complex ores.

A number of determinations of lead in mill-tailings were made in the year 1891; a few of these results are given in Table I.

TABLE I.—*Determinations of Lead in Mill-Tailings by Wet Assay.*

Date. 1891.	On 16-Mesh.		On 30-Mesh.		On 120-Mesh.	
	Weight.	Lead.	Weight.	Lead.	Weight.	Lead.
	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
May 15.....	585.0	1.03	663.5	1.19	330.5	0 89
May 16.....	511.5	1.07	651.0	1.34	374.5	0.80
May 18.....	525.0	0.98	720.0	1.10	475.0	0.74
May 19.....	514.5	1.21	731.5	1 84	468.5	1.16
May 23.....	691.0	2.05	879.5	2.30	370.5	1.44
May 25.....	500.0	1.53	712.0	2.02	397.0	0 89

Date. 1890.	Through 120-Mesh.		Total.		Original Tailings.	
	Weight.	Lead.	Weight.	Lead.	Weight.	Lead.
	Grams.	Per Cent.	Grams.	Per Cent.	Grams.	Per Cent.
May 15.....	118.0	2.62	1647.0	1.200	1647.0	1.22
May 16.....	186.5	3.12	1723.5	1.335	1723.5	1.36
May 18.....	196.5	3.63	1916.5	1.237	1916.5	1.24
May 19.....	199.0	1.62	1913.5	1.481	1913.5	1.50
May 23.....	104.5	3.07	2045.5	2.099	2045.5	2.10
May 25.....	166.0	2.78	1775.0	1.700	1775.0	1.72

* Received October 15, 1904.

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ARTHUR JARMAN, The University, Sydney, N. S.W., Australia (communication to the Secretary*):—I heartily endorse all that Mr. Guess says concerning the shortcomings of the molybdate titration when applied to poor lead-ores. My experience has been that, with ores containing more than 5 per cent. of lead, the results obtained by different assayers working in the ordinary way without duplicate determinations may be expected to agree within 1 per cent. of each other. If the results are important, ores containing less than 5 per cent. of lead require more care and attention than a mine-assayer can bestow on the determination.

However, accuracy is retained, if the following conditions be observed:—a "zero" or "indicator" correction should be applied, the bulk of the solution should be kept small (100cc.), the assay should be kept simmering for some time after it is apparently finished, and the solution should be re-tested before declaring it to be complete.

This claim for accuracy is clearly shown in Table I., which comprises a few results on poor ores given to students who had just completed their first acquaintance with this wet method of lead-assay.

TABLE I.—*Determination of Lead by Various Workers Using the Wet Method.*

Authority.	Sample A.	Sample B.	Sample C.	Sample D.	Sample E.
Demonstrators.	Per Cent. 8.4	Per Cent. 1.6	Per Cent. 2.4 2.1	Per Cent. 4.1	Per Cent. 4.3
Student A.	7.98 8.16				
Student B.		1.39 1.37			
Student C.			2.4 2.26 2.12	4.4 4.45	
Student D.				4.3 4.2	
Student E.				4.08 4.12	4.43 4.38
Mines-Assayer		4.5	4.5		5.5

NOTE.—Students A and B were good workers; C and E, excellent and reliable, and D, reliable but slow.

* Received October 17, 1904.

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The samples noted in Table I. were given to the students with the instructions that 1 g. was to be used for the assay; that duplicate results were to be handed in; that the zero error was to be taken into account; and that the solution was to be kept at the simmering-point for a few minutes after being completed, and then re-tested. The only additional information given was that the lead-content was between 0 and 10 per cent.

In viewing the data in Table I., it is evident that the mines-assayer had no time to spare for such refinements of assay; otherwise, I should not have re-tested the samples before using them, similar assays, but on ores of a higher percentage of lead, being in accord with my own results. It is only with poor ores that the difficulties become a nuisance.

There are a few minor precautions which I always take in making these assays, i.e.—(1) After boiling the precipitate of lead sulphate with water, in order to dissolve out iron and other salts, place the beakers in a tray or dish of cold water for an hour in order to ensure the complete precipitation of the lead sulphate. (2) Wash the precipitate with cold dilute sulphuric acid. (3) Do not finish the washing with water, but dissolve the precipitate at once in boiling ammonium acetate solution. It is then ready for titration as usual. The ammonium acetate should contain a slight excess of ammonia in order to render the precipitate more readily soluble, and also to neutralize the trace of sulphuric acid which remained.

Without these three precautions, I do not consider it wise to assay poor lead-ores or tailings.

In order to avoid paper-slimes, remove the paper by fishing it out from the boiling acetate solution before it has had time to become disintegrated, and squeeze-out the solution that has been absorbed and held in the paper. Filter-paper, 11 cm. in diameter, carries only about 5 cc. of solution, and thus the bulk of it is recovered; then rinse the paper and fingers with water, adding the rinsings to the solution which is now ready for the titration. Except with very rich ores, the quantity of lead remaining in the paper is negligible. The bulk of the solution should be kept small by using 100 cc. of 5 E acetate solution, and never diluting it.

The formation of the precipitate is easily seen during the titration and it is noticeable that the first faint yellow colora-

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tion comes on the test-plate before the precipitate ceases to form. The true end-point, a deeper yellow, appears later on, but it is not sharper than a limit of 2 drops (or 0.1 cc.) of the solution, of which 1 cc. equals 0.005 g. of lead; this means 0.05 per cent. on 1 g. of ore. I am satisfied, when certain of the end-point, that the results are accurate within 0.2 cc. of solution (4 drops), which is equvalent to 0.1 per cent. of lead in the treatment of 1 g. of ore.

The method of precipitation by chromate, and titrating the slight access, is given in Sutton's book,[1] and it is there recommended that carbon disulphide should be used as an indicator because the yellow color of the solution is said to interfere with the starch-iodide end-point.

Since reading the paper by Mr. Guess, I have made a few tests and find that the precaution mentioned by Sutton is not necessary, the end-point being perfectly good and easily recognizable, the finished solution having only a very faint green tint.

I intend to use this wet lead-method in the laboratory, since accuracy can be obtained without special precautions. Ores containing lime are not often met with in Australia, but in assaying foul slags, the lime is troublesome, and the omission of the sulphuric acid treatment should be a substantial improvement of the method.

H. A. GUESS, Cananea, Mexico (communication to the Secretary*): With regard to Mr. Gazzam's experience with the sulphate-chromate method, used by him some years ago in southeastern Missouri, had he omitted adding the sulphuric acid in the treatment of those heavy lime ores (thereby avoiding the formation of the bulky calcium sulphate) he would have found that he could have doubled the number of assays for a given period of time without affecting the accuracy of the results. Since the date of publication of my original paper, I have ascertained by routine work that by using broad 250-c.c. beakers in this modified method, the nitric-hydrochloric solution of the ore, after adding the requisite quantity of strong ammonia and acetic acid, may be readily boiled without danger of foaming,

[1] *Volunteer Analysis*, Sutton, London (1896).

* Received April 3, 1905.

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and the lead may be precipitated therefrom in a thoroughly granular form by a boiling 10-per cent. solution of potassium chromate or bichromate, even if only a few milligrams of lead be present in the solution. This procedure is more expeditious and convenient than agitating the solution in flasks, as previously described and used. Further, except in the case of ores of high iron- or manganese-content, if the solution at the time of precipitation is strongly acetic, the use of 0.5-per cent. of acetic acid in the hot wash-water during the filtration may be omitted. This omission saves much time when making 50 or more analyses at once, since a jet of hot distilled water from a distilled water-stock tank can be used, and the wash-bottle is not needed until the lead chromate is ready to be dissolved in warm dilute hydrochloric acid.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Stock-Distribution and Its Relation to the Life of a Blast-Furnace Lining.

Discussion of Mr. Baker's paper, read at the Lake Superior Meeting, September, 1904.

MR. T. F. WITHERBEE, Durango, Mexico (communication to the Secretary*): Mr. Baker's paper is very instructive in that it shows how furnace-derangements sometimes originate from apparently slight causes.

In 1896 a similar burning through of the lining occurred at the Mayville furnace of the Northwestern Iron Co., Mayville, Wis., in exactly 38 days from the time the furnace was blown in. At that time I attributed it to the fact that the bell was carried several inches to the front side of the furnace, the length of the versed side of the arc described by the lever, thus dumping a little more of the charge on the back side, which made the front side of the furnace-charge more open and consequently increased the circulation of the furnace gases through it.

After reading Mr. Baker's description, I believe that possibly a little more ore was charged on the back side of the hopper, due to the fact that it was nearest to the elevator. This unequal charging, however, was unconsciously done, as the fillers were both honest and faithful.

The illustration of the burned lining of Furnace No. 1, given in Fig. — of Mr. Baker's paper answers very well for the Mayville furnace. It shows that in both cases the destruction of the lining was largely due to faulty construction, in that there were no cooling-plates in the angle of the bosh and in one or two courses of brick above it, where I believe they are more in demand than anywhere else in the furnace. The presence of these cooling-plates at the angle of the bosh would have obliged the current of the hot gases to turn the corner and would have prevented them from following the angle of the bosh completely through the lining and shell.

* Received January 17, 1905.

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The Mayville furnace was hand-filled, but it always showed a tendency to work to the front; so much so that the front tuyere, although not directly over the iron-notch, generally could not be used wide open; most of the time it was bushed down with a clay stopper to 2 in., which served to regulate the condition of the iron-notch.

The accident at Mayville was of minor importance, a spray of water on the brick-work preventing any further damage until 12 cooling-plates could be cast in the pig-bed. A stop of 9 hours sufficed to cut 12 slots through the shell and to set the plates flush with the original inside of the lining at the level of the perforation. Thus repaired, the furnace ran 999 days, and when blown out, the lining was found practically intact on the back side, but at the front side it had been cut through to the limit.

Without questioning Mr. Baker's diagnosis, I would suggest that his description of the behavior of his Furnace No. 1 applies also to a "cored up" furnace, or one that works very stiff in the center, which would account for the cinder and iron remaining around the tuyeres and burning them, and also for the relief which resulted from increasing the volume of the blast, as I understand it, before reducing the percentage of alumina in the slag.

Under such conditions any slag will become viscous from the cooling-effect due to the passage of the comparatively cooler blast through it, the blast never attaining the temperature of slag-fusion.

Another similar instance comes to my mind in the conditions existing at the Calumet furnace, South Chicago, in 1895, where the cinder was always "up;" break-outs high up in the boshes were of constant occurrence, and tuyeres and coolers were frequently burned out. The original dimensions were 18 ft. by 81 ft., with a 14 ft. stock-line and an 11 ft. crucible. There were eight 7-in. tuyeres and a bell 11 ft. in diameter. This furnace had been in blast 4 or 5 months when I took charge, but I was not told that it had "bucked" ever since the campaign was started. Little by little its history leaked out.

At first, the furnace was in blast about 18 months, after which it was idle for almost the same length of time. It was then blown in with no repairs to the lining, and many of the

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cooling-plates projected inward a foot beyond the inner surface of brick-work, but, worst of all, the 11-ft. bell had been replaced by an 8-ft. one.

At the time I took charge the furnace equipment was "down at the heel," 3 boilers out of 12 had been, and were constantly, off for repairs, and the largest blowing-engine was broken down;—adverse conditions which remained during three-quarters of the balance of the campaign (about three months), reducing the blast to one-half the proper volume.

The diameter of the furnace at the tuyeres had been increased to 14 ft., and, combining all of the conditions that then existed, I suspected the presence of a "core," a suspicion which was intensified by the fact that it was impossible to drive a steel bar beyond a point about 2 ft. in front of the tuyeres. By reducing the number and area of the tuyeres used, the one blowing-engine promptly removed the core.

With regard to the quantity of fine coke screened out of the bins, noted by Mr. Baker to be 4 per cent., this was about the loss on Connelsville coke delivered in Wisconsin. I have never had any experience with the Brown charging-apparatus, yet I believe it would have so distributed the materials as to avoid localizing the fine coke. At Mayville all coke was charged that would not pass through a $\frac{5}{8}$-in. mesh-screen.

In small furnaces using very coarse coke, such as Blossburg and Mexican, I have found it a decided advantage to charge dust, provided it was first screened and then charged all but the regularly, in order to have a suitable resistance to the passage of the blast in the furnace. Otherwise the heat would be lost at the tuyeres and "off" iron and other derangements would result.

I am surprised that a slag with only 15.2 per cent. of alumina should have been viscous, and it is with considerable reluctance that I give my experience with aluminous slags. I know that I shall stand almost alone, but not quite so, for there are a few furnacemen who can vouch for the utility of aluminous slags in certain cases. Three times, owing to local and commercial conditions, I have been obliged to use slags of a greater alumina-content than those which Mr. Baker found troublesome. The first time was in making foundry-iron from a rich magnetite, carrying a basis gangue, and requiring a dead flux

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which was supplied by adding 500 lb. of feldspar (labradorite) and 1,500 lb. of magnesian limestone.

A part analysis of the feldspar gave:

	Per Cent.	Per Cent.	Per Cent.
Silica	53.30	54.35	48.27
Alumina	22.00	36.68	15.50
Iron magnetic oxide	7.65		14.15
Lime	5.89	.....	4.20
Titanic acid	.....	.....	8.25

The magnesian limestone had from 2 to 3 per cent. of silica, and between 13 and 14 per cent. of magnesia; the fuel was one-third Connelsville coke and two-thirds anthracite coal, and the blast-temperature was 1,400° F. A very liquid slag resulted from these conditions. The casting-flush was run into "scab-holes," and always carried a heavy back-bone of "kish," while the regular flush ran into the slag-pots of the Kloman cinder-cooler. In cooling down, both the slag-pot cakes and the "scabs" shrunk away from the top crust, leaving a cavity which was studded with beautiful light olive-green, rhomboidal, plate-like crystals, superimposed in a "staggered" order. These crystals measured from 0.25 to 0.5 in. on a side, and were from $\frac{1}{8}$ to $\frac{1}{16}$ in. thick. The fracture was smooth, showing an entire absence of porosity or grain, while the color of the outside of the cake was grayish-green. This slag was so fluid that it ran for hours as fast as it was made into the slag-pots through a Luhrman cinder-notch, having only a 1.25 in. orifice. It was not in the slightest degree viscous, although it contained roughly 32 per cent. of silica and 32 per cent. of alumina.

The second case was at an all-coke furnace, using a mixture of Lake ores and a highly aluminous native ore. Slags from the foundry-iron mixture gave the following analysis: SiO_2, 25.24; 25.32; 28.56; 30.40; 31.18; and Al_2O_3, 25.20; 24.90; 21.28; 20.24; 19.44 per cent.

Magnesian limestone was used and the slag was perfectly liquid, and the sulphur in the iron averaged about 0.02 per cent. The temperature of the blast was 1,100° F., and the fuel-consumption per ton of iron produced was 1,850 pounds.

At the same furnace all Norrie-Day ore was used for 10 days,

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which is known to be highly albuminous (SiO_2, 3.8, and Al_2O_3, 1.55 per cent.).

The daily slag analysis was:

Silica.	Alumina.
Per Cent.	Per Cent.
27.00	20.32
30.40	19.34
21.68	28.12
21.92	28.96
22.10	28.50
26.36	25.58
28.10	24.92
29.72	25.80
28.88	26.80
31.20	25.68

The product was high-silicon, malleable Bessemer iron, carrying also about 0.02 per cent. of sulphur. The blast-temperature was the same as on foundry-iron, and the fuel-consumption was 1,750 lb. per ton of iron produced. This was an experimental run, and there was really no call to add a siliceous ore, as was afterward done, except in deference to public opinion on slag-composition. The fuel-consumption is given to show that high-alumina caused no waste of coke.

The third case noted is the composition of the slag at Durango, Mexico, which varies from 15 to 20 per cent. of alumina and gives low-sulphur, although the blast-temperature seldom exceeds 400° F. The slag runs through a Luhrman cinder-notch having a 1.5 in. opening, and then as far as 80 ft. on the ground. It is not viscous unless lime is short, but it is not so fluid as in the first two cases cited, which may be due either to the fact that it was actually cooler, since the blast-temperature is low, or to the fact that the flux was limestone, while in the other cases a magnesian limestone was used. I do not advance any explanation for this very abnormal result, except to suggest that a highly aluminous slag may be more fluid than one very much lower, especially if combined with magnesia. I believe that the idea of low-alumina and an all-lime flux must have originated with the Bessemer furnacemen, and from their standpoint they are right, but I am sure that an aluminous slag has its advantages under certain circumstances; among others, when making foundry-iron.

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Mr. N. M. Langdon said to me many years ago, when managing the Cedar Point furnace, that "high-alumina means high-silicon,"—a statement which my experience fully corroborated and which has since been confirmed by the German engineer Platz.[1] Hence, for the foundry-iron furnacemen to favor low-alumina is only repeating the mistake of again rejecting the chief stone of the arch. I believe that Mr. Langdon is entitled to the credit for making this important discovery, and I would now add to it, that high-lime means open-grained iron, although occasionally somewhat at the expense of the silicon.

I have known all sorts of undesirable results to be charged to a high-alumina slag, among others that "No. 1 iron (by grain) could be made with it;" but at a certain furnace that idea was at least temporarily exploded by the production of an unusually high percentage of so-called No. 1 iron.

Along in the '80s, the fad of "all-lime, low-alumina" struck the country, and I have seen slags made on that plan with Potsdam sandstone and calcite, which would not run straight ahead the length of the cinder-spout, but would froth up and run sideways indefinitely. Mr. Geo. W. Jamme introduced the use of magnesian limestone in the Champlain district, much to our advantage, as in those days the Bessemer people wanted about 3 per cent. of silicon and a magnesian slag did not have a sticky point, at foundry-iron temperatures. I am confident now that we were enabled to make such high-silicon iron by reason of our high-alumina, though not attributing it to that element.

What I believe to have been an equally unfounded prejudice once existed against magnesian slags that now attaches to high-alumina, mainly on account of its supposed extra corrosive action on the brick-work, but it so happens that that part of the brick-work most exposed to corrosion—the part below the tuyeres—is the most easily and surely protected, even without the use of any cooling-plates whatever.

Alumina does not seem to be at all particular in its preference to unite with silica or lime, hence a refractory slag can be obtained by carrying alumina high, but if the same effect were sought by having an excess of lime, a "lime-set" might result.

[1] Nau in *Iron Age*, January 28, 1904.

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Like Mr. Baker, I should have thought that 0.45 per cent. of titanic acid was partly responsible for the viscous slag, had I not known that Mr. C. H. Foote, while in charge of the Crown Point furnaces, once terminated a blast with ore containing 7 per cent. of titanic acid. He recently told me that he had no trouble with it whatever. I do not know how he fluxed it, but I have an idea that he simply ignored the titanic acid, and considered only the usual constituents of the ore.

When slag containing 32 per cent. of silica and 32 per cent. of alumina was made, alumina was considered a strong base, as was the custom at that time, and so by that theory it was highly basic cinder, but if the same charge had been fluxed, taking alumina as an acid according to present practice, 2,900 lb. of limestone should have been used instead of 1,500 pounds. The iron and titanic acid, present in the labradorite was due to disseminated titaniferous iron-ore, which exists in large bodies in the formation at Lake Champlain, north of Westport, N. Y. The presence of titanic acid in the feldspar was not known, until after its use as a "dead" flux, and so had nothing to do with its selection for that purpose.

I do not believe that fine ore, *per se*, is necessarily so very objectionable in the blast-furnace, but I do believe that derangements occur when it has been improperly charged, or irregularly used. Taking the case of Mesabi ores (which are now used up to 100 per cent., according to Mr. Grammer),[2] if the charging-apparatus permits of no variation in distribution which the conditions of the furnace may occasionally demand, it is quite natural that derangements must occur.

Ores having a varying amount of coarse and fine material should be charged proportionately with as much regularity as possible, and the same practice might apply to coarse and fine fuel, although it might spoil the record of two men charging a large furnace, by adding one or two men per shift to the labor-account, but the avoidance of a single "off cast" per week would largely repay for any extra cost.

When the Cedar Point furnace was first started, a slight irregularity was experienced on account of the varying fineness of the ore, which was delivered to the stock-pile, from drop-

[2] *Trans.*, xxxiv., —.

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bottom cars on a high trestle. The coarse ore would roll to the outside of the pile, and for hours the coarse pieces formed the ore-charge; later, as the center of the ore-pile was approached, the ore-charge was made up largely of fine, and when dumped into the furnace the fine material ran down into the underlying coarse charges, locally over-burdening and under-burdening the furnace, and daily causing a cold spot and a hot spot. This bad feature was overcome by placing screens under the track and separating the coarse ore from the fine, which allowed the two kinds to be charged in relative proportions. In this case only one kind of ore was used.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Improvements in the Mechanical Charging of the Modern Blast-Furnace.

A Discussion of the Paper of David Baker, read at the Lake Superior Meeting, September, 1904.

(Washington Meeting, May, 1905.)

MR. JOHN J. PORTER, Chicago Ill. (communication to the Secretary*):—Mr. Baker's account of his experiences with stock-distribution has been particularly interesting to me, as it confirms the opinions which I had formed concerning the rapid failure of linings in several furnaces that have recently come to my notice.

These furnaces, four in number, are of modern construction, having been built or remodeled within the past two years, and are equipped with a well-known type of skip-hoist and double-bell charging-apparatus. In every case, a hot spot has developed within a few months after blowing-in, which has already compelled relining in two cases, and the probabilities are strongly against a long campaign in the other two. Since the hot spot in each case appeared in a position approximately opposite to the skip-hoist, and since the sizing, mentioned by Mr. Baker, is very apparent, it would seem as though there could be no doubt concerning the cause of the trouble. Nevertheless, there are some who cannot see it in that light; for the managers of two of these plants, at least, do not admit the existence of bad distribution, and have satisfied themselves by putting in cooling-plates, thickening the lining of the in-wall, and changing the brand of the fire-brick used for the construction of the walls.

From observations of the dumping-action of the skip-hoist, and from subsequent inspection of the stock-distribution in the furnace itself, I arrived at the conclusion, some time ago, that no system of filling could be perfectly satisfactory where the stock is dumped into the hopper from one side, also that some

* Received December 16, 1904.

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form of rotary hopper or other circumferential distributing-device must eventually succeed the present form of apparatus. It is therefore with some satisfaction that I note the present tendency to criticize the double bell-and-hopper arrangement, and to give increasing attention to the Brown distributor.

In connection with this latter device, it has occurred to me that the construction might be somewhat simplified by substituting for the side opening and door on the revolving-hopper a central opening closed by a small bell, suspended in such a manner as to be free to rotate with the hopper. I have not, however, attempted to work out the details of this arrangement, but merely offer the suggestion in case any one else should care to do so.

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The Equipment of a Laboratory for Metallurgical Chemistry in a Technical School.

Discussion of a Paper by Mr. C. H. White, read at the Atlantic City Meeting, February, 1904.

(Annual Meeting, February, 1905.)

ARTHUR JARMAN, Sydney, New South Wales, Australia (communication to the Secretary*): All designs for modern metallurgical and chemical laboratories should provide each student's desk with a hood or cupboard for the removal of fumes, but I doubt the advisability of placing air-baths and similar apparatus close to the hot plate. It seems to me that unless the draft be exceptionally strong the fumes from ordinary wet assay work will corrode the iron-work of the fittings, in which case the baths would have to be removed to a general desk in order to lighten the work of the laboratory attendant. Since the laboratory described by Mr. White has been in use for two years, I should like to ask if any inconvenience has resulted from this corrosion. If not, it is certainly an important improvement to have the air-baths and similar apparatus placed on the student's desk.

The hoods described by Mr. White are supplied with a good draft by means of a fan, which gives them a great advantage over those at the metallurgical laboratory at the University, Sydney, in that no doors to the hood are needed. Each hood in the laboratory at Sydney is connected by a glazed earthenware drain-pipe with a vertical flue, 20 ft. high, having a lighted gas-jet at the bottom to assist the draft. Although the draft is good, fumes of sulphuric acid cannot be drawn off unless the doors of the hood are lowered more than half-way. In my opinion, a fan should be used for the purpose of creating the draft, and glazed drain-tile should invariably be used for the flues because no other material possesses the same advantages.

I know of a recent instance in which a laboratory containing

* Received October 17, 1904.

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about 20 desks has been fitted with flues of sheet-iron, tarred inside and kalsomined on the outside. How long these will last is a question, but from personal experience with flues of this character I should say that they will crumble to pieces within three years.

The floors of the hoods of the University laboratory at Sydney are covered with 6 by 6 in. white glazed tiles, ¾ in. thick, set in asphalt directly on the top of the desk. Should the desk-top become warped by the heat from the burner and hot plate, the tiles may be disturbed, although in seven hoods after four years of service only two tiles had become sufficiently displaced to allow acid to make its way through the joints. A foundation that will not buckle when heated is a needed improvement, and for this purpose cement would answer, the tiles being set in a small quantity of asphalt in order to prevent acids from attacking the joints. Since Mr. White does not specify the method of placing the tiles in the laboratory at Harvard, I should like to ask what construction was finally adopted.

With regard to the means used to furnish exhaust and pressure, each desk in the laboratory of the University at Sydney, accommodating four students, is equipped with a water suction-pump which gives, according to tests with a mercury-gauge, a 680-mm. vacuum or an actual pressure of only 80 mm. Occasionally, better results than these have been obtained, but pumps of this kind have the inherent defect that rust from the water-supply pipes will clog them after a long period of inaction. The use of a blowing-engine to furnish both suction and air under pressure removes this difficulty and also avoids the variation in pressure which inevitably accompanies the use of water-blowers or suction-apparatus.

At Sydney much difficulty was experienced in obtaining a satisfactory lock for the desks. Finally we adopted a circular brass plate which covers the edges of the doors, and is firmly secured by a pad-lock. Other locking-devices can often be removed by the use of an ordinary screw-driver.

Mr. White does not mention the detailed construction of the sinks in the laboratory at Harvard. The usual lining of sheet lead becomes quickly corroded by acid and other liquids, and at Sydney the cost of plumbing repairs became large. In order to reduce this item, Mr. Schofield, Demonstrator in

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Chemistry at the University at Sydney, introduced the use of a false bottom, over a layer of broken marble. The false bottom was first made of perforated sheet-lead, but as this was rapidly eaten away, it was later replaced by glazed earthenware trays. The use of marble under a false bottom has saved much trouble and has considerably reduced the cost of plumbing-repairs at Sydney.

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The American Mining Engineer.

Discussion of the Paper of Albert R. Ledoux, read at the Atlantic City Meeting, February, 1904.

ARTHUR JARMAN, Sydney, New South Wales, Australia (communication to the Secretary*): Some remarks upon Dr. Ledoux's interesting paper from one who has not visited America may, I hope, be of interest to members. It appears to me that there is considerable difference between the training of the American and of the British mining engineer. Not that the subjects are different, but rather in the way in which they are handled, the illustrations being, as far as possible, mining in character.

Colloquial instruction is unknown in English curricula, but could be introduced with advantage. Students here assure me that an hour of conversation is of far more use to them than an hour of the best lecturing. In England a school-year of 6 or 7 months would be considered very short indeed; but Americans do not consider it so, for the reason that the rest of the year is mostly spent in acquiring practical acquaintance with their subjects in works or in the field, and the shorter courses, when coupled with this large amount of outside experience, produce an essentially practical-minded engineer, who is also more of a business man; whereas the English system, though excellent from a scholastic point of view, produces the theoretical rather than the practical man.

British schools are waking up, however, and the practical aspect of the engineer's training is receiving greater consideration. Once launched upon his career, the American engineer is immediately called upon to exercise his self-reliance and ingenuity, if he is to retain the confidence of his employers; but what can we say of English directors? In how many cases is the engineer allowed to guide the destinies of his company? Witness several recent cases of gross waste of money upon unsuitable plants, and the perpetuation of prehistoric methods of

* Received October 17, 1904.

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smelting. What would Americans say of a copper-works whose directorate refused to sanction, in 1897, the use of water-jacketed cupolas, or bessemerizing plant, and insisted upon the retention of brick cupolas and reverberatory roasters? An American directorate would carry out the alteration while the other was thinking about it. Again, with respect to the "interchange of ideas," one may almost say that it does not exist in Britain. The incident cited by Dr. Ledoux of the copper-bessemerizing works in South Wales is typical of the attitude adopted by many English directors, and is not merely an isolated instance.

I personally made special trips to three large works in Wales, in the hope of seeing electrolytic copper-refining; and in no case was I permitted to view the works.

What could an American school accomplish in such an environment? It would immediately lose its principal and distinguishing characteristic, namely, its connection with modern smelting-works. It is this which enables the American schools to turn out the men that they do; and the rapid progress afterwards made by such graduates seem to me to be due to the necessity for initiative, which is recognized on the American side of the Atlantic much more than upon the other.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Comparison of Methods for the Determination of Carbon and Phosphorus in Steel.

(Washington Meeting, May, 1905.)

A discussion of the paper by Messrs. Jüptner von Jonstorff, Blair, Dillner and Stead, read by title at the Lake Superior meeting, but presented first at the New York meeting of the Iron and Steel Institute (October, 1904), and here published under a mutual agreement between the two Institutes.

Correspondence.

MR. R. HAMILTON (Glengarnock) wrote to say that it was known for many years, that unless certain precautions were taken in dissolving the sample, there was a liability to obtain a solution from which the ammonium-molybdate solution would not precipitate the whole of the phosphorus. In 1877 Blair and Finkiner independently attributed this interference to the presence of carbonaceous matter in the solution, derived from the action of acids on the combined carbon of the metal, and they directed that the solution should be prepared in such a manner as to destroy these carbonaceous compounds. This was practically effected by evaporating the acid solution to dryness and roasting the residue. Adolf Tamm, in 1884, accentuated the same view. There are methods founded on the above-mentioned observations which leave nothing to be desired in point of accuracy, but the evaporation to dryness with subsequent roasting and re-solution consumes much time, besides the minor inconvenience caused by large amounts of acid fumes. In 1888 P. W. Shimer suggested that the incomplete precipitation of phosphorus was due to incomplete oxidation of the phosphorus, and proposed to render all the phosphorus precipitable by oxidizing the solution with potassium permanganate, etc. Experiments were made to test the accuracy of this view (details will be found in the *Journal of the Society of Chemical Industry*, vol. x., p. 904), and the results obtained seem to prove that it is the correct one.

It was first proved that carbonaceous matter in solution did not prevent the precipitation of properly oxidized phosphorus

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as ammonio-phospho-molybdate. This was done by using a pig-iron low in phosphorus, but high in combined carbon, and adding known quantities of a solution of phosphoric acid to it; this addition was made both before and after the solution of the iron in nitric acid, and in all cases the whole amount of phosphoric acid was obtained, though the solutions were not evaporated to dryness, in some cases, indeed, only to half bulk. The next step was to find if the phosphorus in iron could be rendered completely precipitable by oxidation while in solution. For this purpose there was used a sample of white pig-iron containing 2.54 per cent. of combined carbon and 1.36 per cent. of phosphorus. By dissolving as usual, evaporating to half bulk and precipitating, without any attempt at oxidation, two-thirds of the phosphorus was obtained; the filtrate from the yellow precipitate was worked with to induce further precipitation, but no more came down. In a supplementary experiment the remaining one-third of the phosphorus was obtained in the filtrate from the yellow precipitate by oxidizing the said filtrate. Several methods of oxidation were tried, details of which will be found in the original paper; the method found to give the best results was to oxidize the permanganate and remove the resulting precipitate with hydrogen peroxide. By this means the whole of the phosphorus was obtained, and that in a very short time. The iron was dissolved in nitric acid of 1.135 or 1.2 specific gravity, oxidized twice with permanganate (removing the precipitate with hydrogen peroxide) neutralized with ammonia between the two oxidations, the flask stoppered and shaken vigorously for five minutes, cooled and filtered straight off. The phosphorus was weighed as magnesium pyrophosphate, and the precipitate proved free from silica. The whole of the phosphorus was also obtained by dissolving the iron in hydrochloric acid with addition of bromine.

The following is the summary of the original paper: "I think it may be fairly assumed that the two sets of experiments described prove to satisfaction (1) that the carbonaceous matter in a solution of iron does not prevent the complete precipitation of properly oxidized phosphorus as ammonio-phospho-molybdate; (2) that the reason of the incomplete precipitation of phosphorus from a nitric acid solution of iron, which has not been evaporated to dryness and roasted, is that the phosphorus

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has not all been oxidized; (3) that in such solutions all the phosphorus can be rendered completely precipitable, in the wet way, by means of certain oxidizing agents."

Though it is thirteen years since the paper was read, and though the method described presents many advantages, the writer finds that it is not generally adopted. He has made these notes with the double object of giving credit to those who made the discovery of the important influence of permanganate, and of drawing attention to the fact that their observations have been fully confirmed.

MR. E. H. SANITER (Rotherham) wrote referring to Mr. Gunnar Dillner's remarks on the direct combustion of ferro-chrome for carbon, and pointing out that it might be interesting to state that by mixing the ground ferro-chrome with oxide of copper containing 25 per cent. of litharge (PbO) the whole of the carbon was burnt off in thirty minutes.

MR. STEAD, in reply to the correspondence, said he, as one of the committee, welcomed the remarks of Mr. Hamilton, as they show how important it is that the phosphorus should be fully oxidized to P_2O_5. He (Mr. Stead) had not found it necessary to use permanganate when strong hydrochloric acid was employed for dissolving steel, but if weak nitric acid were used, the phosphorus was not fully oxidized, and it then became necessary to use it. In regard to Mr. Saniter's remarks, he quite agreed that the method he described would give correct results.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Acid Open-Hearth Manipulation.

(Washington Meeting, May, 1905.)

A Discussion of the paper by Messrs. Andrew McWilliams and William H. Hatfield, read by title at the Lake Superior meeting, but presented first at the New York meeting of the Iron and Steel Institute (October, 1904), and here published under a mutual agreement between the two Institutes.

Correspondence.

MR. J. J. MORGAN (Workington) hardly thought that the results obtained were sufficient to warrant the conclusion arrived at by the authors. The variations in the silicons of the bath samples were so small that one might reasonably doubt whether they were really due to the causes assigned, or to:

(1) The non-uniformity of the bath. A 25-ton bath of liquid metal was not a homogeneous mass, and he had found samples taken simultaneously from different parts of the bath to vary—especially in silicon-contents; in some extreme cases the silicon being nearly twice as high in one part as in another.

(2) Intermixed slag. No doubt every precaution was exercised in sampling, but if taken in the usual manner by dipping a sample spoon through the slag into the metal underneath, it was a difficult matter to obtain a sample altogether free from slag.

(3) A combination of 1 and 2.

If by the statement, "This with other results from the experiment, bearing out the general trend of trials made with other ends in view, and hence not followed out in detail at the time, was useful in the making of certain special steels for commercial purposes," the authors referred to the manufacture of high silicon steel, he would remind them of the well-known fact that the influence of silicon left in, or unremoved from, steel was much more injurious than that of the same amount of added silicon. From the fact that the silicon was eliminated from the bath and went to slag, and, under the special conditions of working, again entered the metal, it might be argued

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that the silicon was added. Granting this, it remained to be proved whether silicon thus "added" behaved in any way differently to silicon left in, as generally understood, in the metal.

Mr. E. H. Saniter (Rotherham) could not see that the authors' contention was confirmed by their experiments, because in Table I., red brick charge, before adding lime and ore, the silicon was 0.049 per cent., and after 0.055 per cent., with a less siliceous slag. It was also interesting to note that while the addition of 1 cwt. of lime raised the lime in the slag 0.72 per cent., it raised the magnesia 1.48 per cent. Again, in regard to Table II., lime charge. Before adding the lime the silicon was 0.049 per cent., with 55.45 per cent. silica in the slag, and after adding the lime it was 0.052 per cent., with only 53.54 per cent. silica in the slag. It was to be noted that the words lime and limestone appeared to be used indiscriminately. The same peculiarity was to be observed with the magnesia charge, Table III. Before adding the magnesia the silicon was 0.034 per cent. silicon, with 55.72 per cent. silica in the slag, while after it was 0.037 per cent., with only 54.03 per cent. silica. These differences in silicon were very small to argue about, but such as they were, they were directly opposed to the authors' contention. In conclusion, he thought that the addition of 8 cwt. of red bricks, which would float about in the slag, and be rapidly melted by the gas, would practically have no cooling effect on a 25-ton bath of steel. He suggested that the experiment be repeated, with the addition of 8 cwt. of heavy steel scrap low in silicon, which would reduce the temperature of the 25 tons of molten steel about 30 degrees centigrade.

Messrs. McWilliam and Hatfield, in replying, expressed their indebtedness to Messrs. J. J. Morgan and E. H. Saniter for their courteous criticisms, as they were always much interested in the opinions of others on any open-hearth research work. Although Mr. Morgan was quite correct in saying that the differences in silicon were small, they could not agree that they might be due to inclusions of slag or errors in sampling owing (1) to the care with which the samples were taken, and (2) to the similarity in the trend of results in former cases

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where many experiments were made on one particular line. Had Mr. Morgan's last two sentences formed part of a paper instead of a criticism they would have called for proof of his alleged well-known fact that silicon left in was more injurious than silicon added to the metal in the ordinary way. In Mr. Saniter's criticism there were some misstatements, a misunderstanding and an opinion differing from that of the authors. In the red brick charge there was no 0.055 silicon, and if the 0.053 silicon was meant the composition of the corresponding slag was not given, and with regard to the real point which seemed to be troubling him, if he remembered this action of a slag was not instantaneous (the rate under defined conditions was shown in former work), the matter would be quite clear to him. The manganese was not shown as increased by the addition of lime, as he stated. Mr. Saniter's remarks on the lime and magnesia charges were curious, as he disagreed with what was not stated, and pointed out something that was fully dealt with in the paper on p. 213. Few practical men connected with acid Siemens work would agree that 8 cwts. of cold red brick would not cool even a 25-ton charge, and as the authors had fully convinced themselves on the matter referred to, they did not propose to try the suggested experiment.

In conclusion, for the sake of students, the authors would point out that their 1902 paper should be read before this one, that their opinions on the influence of the basicity of a normal type of slag on the silicon-content of the steel were the result of very many experiments; that the numerical results on lime slags were from one special heat only, but were in accord with certain single tests made before; and that the magnesia and manganese dioxide results, though obtained from only one special heat each, illustrated points of theoretical interest which would be appreciated by any careful thinker.

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[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

The Development and Use of High-Speed Tool Steel.

(Washington Meeting, May, 1905.)

A Discussion of Mr. J. M. Gledhill's paper, read by title at the Lake Superior meeting, but presented first at the New York meeting of the Iron and Steel Institute (October, 1904), and here published under a mutual agreement between the two Institutes.

Discussion.

MR. E. WINDSOR RICHARDS, Past-President, said that at a recent meeting of the Institution of Mechanical Engineers a paper was read on tool steel, and he called the attention of the members to the high-speed tool steel at the Bethlehem Works. They would have the pleasure of seeing those wonderful works and would see 60, 70, 80 and 100 tons on lathe with half-a-dozen of these high-speed tools running.

PROF. H. M. HOWE (Columbia University) said that they all looked upon these high-speed tool steels as of very great importance, particularly to the machine shop. One point occurred to him: Had it really been shown that Swedish steel is necessary, or better than equally pure steel that is not Swedish; and is cemented steel really necessary, and is there any advantage in using that rather than equally pure open-hearth steel? It might be well if the author could indicate the kind of evidence on which he bases his assertion that it is necessary to have cemented Swedish steel. Many were hoping for the development of high-speed steels from elements which exist in great quantities, such as silicon, manganese and chromium, rather than from rarer metals like molybdenum, vanadium and tungsten, the supply of which is likely to be soon exhausted. If the high-cutting speed was to last more than a generation, it seemed to him that they would have to draw upon the common elements and not upon the rare ones.

MR. TOM WESTGARTH (Middlesbrough) said he fully appreciated what had been said about the advantages of using high-speed steel, because great benefit had been derived from the

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invention in the works of his firm, and he desired to give his thanks to the American engineers who commenced the investigations which led to the discovery and use of high-speed tool steels. While he agreed with the very great advantages arising from the use of high-speed steels, he thought the author of the paper had, no doubt unwittingly, overlooked the fact that part of the credit was due to the improved machines which had concurrently come into use. When his firm commenced using high-speed tool steel it was very soon found that the capacity of machines built to ordinary designs was reached, and therefore it became necessary to get new tools or alter and strengthen existing machines, and thus a means was obtained for using the special tool steel to every advantage and the output was greatly increased. He therefore thought some of the credit must be given to the new and altered machinery. He observed on the table some samples of work labelled with the time occupied when using new and old steel, and he thought the comparisons were a little misleading, because the whole of the time saved was not due only to the use of steel, but of special machines, and besides, he thought some of the times given for the old kind of tool steel were unduly long. An interesting development occurred in his firm's works when it recently became necessary to build a new smith's shop. It was found that although their trade was increasing a smaller smith's shop could be used, because of the increasing amount of work done in special machines and by the use of high-speed steels. In connection with using high-speed steel an interesting discovery was made by one of his workmen, who found that when roughing forgings and iron or steel castings he could obtain a considerably better result by first painting the work over with a chalk mixture, such as that used for marking off.

Mr. H. H. Campbell (Steelton, Pa.) said the point raised by Prof. Howe that Dannemora iron was necessary should be carefully considered before it was finally recorded, for it involved the question whether it was necessary in the manufacture of any tool steel to have Swedish iron in order to produce the best results. The traditions of the past should not be accepted as conclusive if there is any way of testing the matter. His own company made these special steels, and while they did not

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think they made them better than anybody else in the world, they believed they had made as good a product as several other makers, although they did not use Swedish stock of any kind. It would be interesting to have proof that Swedish iron was necessary, and important to know the reason why.

Dr. John A. Mathews (Syracuse) said: It has been the experience of the company which I serve that Swedish iron is not necessary. We get equally good results using Swedish or a great many of our domestic irons which in analysis are as good or better than Swedish iron. Sulphur 0.007 and phosphorus below 0.01 are not unusual in American irons. The tendency in regard to high-speed steels has been to force one grade of high-grade steel into use for all grades of work. We shall have to differentiate a little when we come to a better knowledge of alloy steels. We undertake in the case of carbon steels to select the steel according to the use to which the steel is to be put, and we shall soon have to do the same with high-speed steel. There are a great many alloy steels that will do remarkable work, but they cannot be produced economically or commercially. We must strike an average between what we can make economically and what can be used to best advantage. We have used some hundreds of tons of molybdenum, and could supply the date Mr. Gledhill's paper lacks concerning this metal. I think that it is equal to tungsten when properly handled. We have probably made as many or more trials than Mr. Gledhill in all sorts of combinations of tungsten, molybdenum and chromium, and the conclusions that Mr. Gledhill has put before us are practically confirmed by our own experience. We appreciate very highly that he should give us such information, for there are not many manufacturers who would put this before the public.

Mr. A. S. Pye-Smith (Sheffield) said: Being one of those who are at present engaged in the manufacture of the original Mushet steel, I have naturally taken considerable interest in this paper upon high-speed steel. I do not know, however, that I have much to add to what has been said. The paper is extremely interesting, and I think the makers of high-speed steel will be grateful to Mr. Gledhill for giving so much infor-

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mation to the users as to its proper use and treatment. The maltreatment of good steel by the users is one of the great troubles the steel-maker has. He has it with the old crucible steel still, but I think this paper will considerably help those who use high-speed steel to treat it properly. There is one point I should like to speak of, viz., that the steel being so hard and brittle, it is very important to avoid sharp corners or angles as far as possible in making tools of it. Further, many tool-makers stamp on their tools some letter or other mark, sometimes stamping them rather deeply. These marks should not be put on parts of the tool that bear any strain, as they form the starting place for fractures just like a diamond scratch on a piece of glass. I have seen several tools broken across where they were stamped just in this way. With reference to the influence of the various elements, Mr. Gledhill has not made any mention of the influence of manganese on high-speed steel. Mr. Metcalf, in his "Manual for Steel Users," lays great stress upon the importance of manganese as the hardening element in self-hardening steels. I should like to know whether Mr. Gledhill has carried out any experiments to ascertain whether manganese is detrimental to or improves high-speed steel. With regard to molybedum, my experience seems to be the same as that of Dr. Mathews, namely, that manufacturers have used molybdenum on this side of the Atlantic much more than on the other. As to the use of the purest qualities of Swedish or Dannemora irons, I recently saw it stated in *Engineering*, under the trade report from Sheffield, that, as the best Swedish irons did not enter into the composition of high-speed steel, the trade in those brands was not very brisk.

Prof. J. W. Richards (Bethlehem, U. S. A.) said: The electrical heating of the tool is an adaption of the well-known Burton water-pail forge. The current sets free hydrogen and the arc jumping across the sheath of gas causes local heating. The mention of the use of molybdenum brings to my mind the difference between the practice in Europe and America. On looking over that very fine work on steel recently written by Mr. Harbord, the word molybdenum is not to be found in the book, and the statement just made by Dr. Mathews that several hundred tons of molybdenum have been used in America, would

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tend to show the differences in the practice in the two countries. Mr. Pye-Smith says that high-speed tool steel is very hard and brittle; and in Mr. Gledhill's paper the influence of the carbon in increasing brittleness is emphasized, it being pointed out that from 0.4 per cent. to 0.9 per cent. is the most advantageous quantity of carbon to be used if brittleness is to be avoided. The particular advantage of using molybdenum is that it gives all the desirable quantities (*sic*) of high-speed steel, with carbon kept below 1 per cent. So that the desirable qualities are obtained by the use of molybdenum and quite low carbon, thus avoiding the hardness and brittleness caused by the higher carbon used in high-speed steels having no molybdenum—the ordinary high-speed tungsten steels.

MR. OBERLIN SMITH (Bridgeton, N. J.) said: I am very much interested in these steels and I have made experiments with them. There is no doubt that the steel-makers have opened an entirely new era which will bring in more improvements. I am not a steel-maker, nor a chemist, but I am interested in the mechanical side. I am using a new shop which has nothing in it but high-speed. The change in machine-tools is going to do more than we think or realize at present. With such tools as lathes and milling-machines we should not see much change without an enormous increase in strength. We have to avoid the vibration owing to the higher speed. I do not think that we want heavier spindles, but heavier beds to give more stiffness than we have been used to. Now, other tools of the reciprocating class are destined to be revolutionized, especially planing tools. We can increase the speed of rotative tools to any amount. But in the use of the reciprocating tools, the inertia soon imposes a limit. We cannot use these high-speed tools in certain cases. In spite of all that we can do we cannot get up to from 100 to 200 ft. per minute. Therefore one of the great changes that will come over us in a very few years will be with planers, moving tools on stationary tables and works. This is one of the revolutions that is bound to come in machines. Another one is to abolish planers altogether. In my opinion the development of the milling-machine has only just begun, and there are going to be wonderful strides in substituting it for the planer. We shall not only get the much

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higher speed that is possible with using these high-speed tools, but we will get more work and have fewer strains. We have the advantage, too, of some of those rotating tools which pass round. I find in equipping a whole shopful of tools that, with the changes due to both the high-speed steels and all the conditions necessary to motor driving, very interesting problems arise. It will certainly require great ingenuity on the part of the high-speed tool-makers.

Mr. W. Carter (Manchester), replying in the absence of Mr. Gledhill, said: Mr. Westgarth says that the high-speed steel has condemned many of the existing machine tools, and only by building special machines is it possible to get the best out of the high-speed steels. This is quite true as far as my experience leads me. But still I wish to say that I have proved that high-speed steel is economical even when applied to the oldest machines that are usually found in most workshops. All our planing machines that were formerly cutting at 18 to 24 ft. per minute are now cutting at 35 to 40 ft., and that has been obtained by merely putting a larger drum on the line shaft. Considering that the cost of a tool would be, say, 7s., it would not be necessary to work very long at the increased speeds to pay for the high-speed tool. The orders are that all machines are to be equipped throughout with high-speed steel, it having been proved that it is profitable to do so. I do not think you could have a stronger advocate for building special tools than Mr. Gledhill. You will find that he makes a special point of it, and in the Openshaw Works are installed some of the most powerful machines yet built for rapid cutting. Mr. Westgarth's remark about the smaller smith's shop bears out the author's statement that the introduction of high-speed steel must have a great effect upon reducing the expenditure on capital account. Referring to the samples, I may say that the intention was to get a comparison between an absolutely modern method as against a method that was in vogue previous to three years ago, and that the times as stated are absolutely the net times. The two articles were made by competent operators using ordinary tool steel, and the times given include the centering and the necessary stretching. The times are right to the minute. Dr. Mathews of the Crucible Steel Company of

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America, I understood to say, was not satisfied that one brand of steel was capable of universal application. Perhaps that was not so, but so far as we have gone at present it is found that for most purposes, with all its defects, the high-speed steel wherever applied beats out of service ordinary steel. Whether that will remain so as the high-speed steel is further developed it is not very easy to say; but at the present time that is generally found to be the case. Mr. Pye-Smith made some interesting remarks, and I can fully bear out what he said about the sharp corners and deep stamping. As one travels round one meets some users of high-speed steel who seem to wish to bring about its death, and they put it to tests that they would not dream of putting ordinary tool steel to, and if it breaks they say that the high-speed steel is no use. I may say that some months ago we tried an experiment to prove whether high-speed steel would stand the loads that would be met with in ordinary workshop practice, and for that purpose a tool 1¼ in. square was used. It was used in a 40-in. lathe and we took a cutting with that tool ½ in. deep, and ⅛ feed. Since the load on the tool is independent of the speed, the speed does not matter, but we got the maximum power obtainable from the lathe, which gave a speed of about 25 ft. per minute. The cuttings are at the end of the room, and they prove without any doubt at all that if high-speed steel is applied with reason and common sense it will not fail to give satisfaction. Mr. Oberlin Smith thought the machines ought to be heavier and more rigid. We are making them so. But I cannot see eye to eye with him in regard to the planing machines. I have seen a 7-ft. planing machine running at 65 ft. per min. and returning at 160 ft. taking a cut ½ in. deep by ⅛ feed. Now cutting at 65 ft. with the feed mentioned will try the best of high-speed steels. The work since speeding up to 65 ft. has been produced at 40 per cent. of the price previously paid. In my general experience in machining articles of short length, milling is usually quicker owing to the time spent in reversal of the planing machines being such a large portion of the total time taken. But where the articles are of any length the planing machine is found to be a long way ahead of any milling-machine yet built. I could give the names of many firms who are using both systems. Concerning the question of a planing

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machine having moving tools, it will be interesting to state that many such machines are at work at Openshaw. In fact, the whole of the armor-plate planing machines are built on those lines, and we have as many as sixteen tools cutting at one time. The tools are fixed in separate tool-boxes mounted upon the cross slide which is carried by two uprights which receive the motion from two driving-screws. The plates do not move at all, and the machines are constructed to cut both ways. I do not know whether machines of similar design have been constructed for light work, but machines, as described, have been built, for heavy work, for some years past.

Correspondence.

DR. H. C. H. CARPENTER (Teddington) wrote that Mr. Gledhill was to be congratulated on the publication of so valuable a paper. As the manufacture and heat treatment of such steels are still in the experimental stage, it might be of interest to state that for the past eight months he (Dr. Carpenter) had been engaged in his capacity as a Carnegie Research Scholar in investigating their behavior under varying thermal treatment, and proposed to submit the results of his experiments to the Institute in due course.

MR. JOHN LITTLE (Sheffield) said it appeared to him that relatively far too much importance was placed on the feed and speeds at which high-speed steel would work, and not nearly sufficient attention was given to the question of the economy obtained by greater durability. As one of his firm's representatives once remarked, a great difficulty in selling high-speed steel was owing to the name of "high-speed." In many cases engineers only seemed to think of this one aspect, and they often pointed out that theirs was special work, with perhaps a machine working only 25 per cent. of its time, the remainder of the time being occupied in setting, etc., and that, therefore, if the work were done twice as quickly it would only save them a comparatively very small amount, and they therefore failed to see the advantages of the extra expenditure. This really was not the question. The economy arises: (1) Because a man does not have to take his drill or milling cutter out of the machine to be ground, so that he saves some of the

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time the machine would otherwise be standing; (2) because the drill, being more durable, does not wear small in diameter, and consequently does more accurate work, as it continues to fit the bushed hole in the jug; (3) because the fewer times that a drill is ground the longer must be the life, as grinding of course shortens the drill each time. As practical illustrations of the foregoing, he stated that in one department in his firm they manufactured a large number of milling cutters. A certain customer was in the habit of placing a large order every January at a time when they were supplying mill cutters of ordinary steel. At last they persuaded the firm to use their 0172 High-Speed Steel. The usual order did not come the following January, and their representative went to inquire about it. He was taken into the stores and shown more than half of the cutters which had not even been used, as the others had lasted so much longer. The firm informed him that they did not expect to order again till the following January, and next year they would not order more than half the quantity, as one year's supply was sufficient stock. So far he had only dealt with initial cost; there was also the saving in grinding and the time the machine was not occupied, men's time going backwards and forwards, use of grinder, etc. He would give them another example: A large firm who used their goods made a rule that the men were not to grind their own tools, and kept a special department for that purpose. This department consisted of seven men. They adopted high-speed steel throughout, and now that department consisted of two men, who could easily cope with all the work. Again, a firm of factors who handled his firm's goods made a calculation when they started as to the number of drills they had been in the habit of selling. When they had high-speed drills they found the demand as regards the number of drills was less than half on account of the extra durability. It was not too much to say that the extra price of high-speed steel was saved on the cost of emery wheels alone.

Mr. F. M. Osborn (Sheffield) pointed out that while Mr. Gledhill had referred to the discovery of self-hardening steel by Bobert Forester Mushet, Bessemer Gold Medalist of the Iron and Steel Institute, he had made no mention of the influ-

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ence of manganese in high-speed steel. The importance of manganese as a constituent of steel seems to have influenced Mr. Mushet in his experiments. While he was experimenting in 1868, not with a view to producing a cutting steel, but an entirely different object, he found that one of his trial bars had the property of becoming hard after being heated, without the hitherto necessary water quenching. The element which gave the steel this property was tungsten. Mr. Hadfield fully explained the influence of this element in his recent contribution. In 1871 the Mushet process of making steel was taken up at the Clyde Steel Works, Sheffield, under the direction of Mr. R. F. Mushet and his two sons, and since then has been introduced into every engineering workshop in the world. Mushet steel was generally called self-hardening; but it was discovered by several engineers independently, of whom it is believed the late Mr. Henry Gladwin was the earliest, that a much better result was obtainable if the cutting portion be reheated and cooled in an air-blast. Tools which had been laid on the ground and cooled by the draught of air from under a door led to this discovery. It was then proved that tools which were reheated to a full scaling, or almost yellow heat, did better work. It was difficult, however, to get the smiths to do this, and in the days before the introduction of high-speed steels, principals and managers took little interest in the heat treatment and work of cutting tools. The following which appeared in the September issue of the *Iron and Steel Magazine* was of interest in this connection:

"In a paper on 'Alloy Steels,' read by Mr. William Metcalf, he draws a sharp distinction between ordinary self-hardening or Mushet steel and those special steels now known as high-speed steels. 'The present high-speed steels,' the author writes, 'are in no sense of the word air-hardening.' And we find this opinion shared by other metallurgists and engineers. To us it seems, however, that no such distinction can be made between these two varieties of steels. Mr. Metcalf himself says that the discovery of high-speed steel resulted from the overheating (possibly accidental) of some ordinary self-hardening steel. This treatment, which had hitherto been considered ruinous when applied to self-hardening steel, resulted in imparting to it those wonderful high-speed qualities now so well known. By

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this treatment the self-hardening steel was converted into high-speed steel. We want no more conclusive evidence of the fact that no sharp demarcation can be drawn between self-hardening and high-speed steel. It may be that, as at present manufactured, the high-speed steel differs materially in composition from Mushet self-hardening steel. It may be that the proportion of tungsten has been increased and that of manganese lowered, or that in some brands the tungsten has been replaced by molybdenum. The fact remains, nevertheless, that to all tungsten steels of the self-hardening variety, if properly treated, high-speed qualities may be imparted. By altering the composition, as pointed out, these high-speed qualities may be intensified, but the difference between the two varieties of steels remains at best one of degree, not of kind."

This is exactly the case regarding the steels manufactured by the writer's firm, viz., R. Mushet's Special Steel (the original self-hardening), commonly called "Mushet Steel," and Mushet High-Speed Steel. The latter is made on the same principles, and its present form is the result of some two hundred different trial mixtures, and hundreds of lathe trials cutting various metals. The hardening is best done by reheating tools after they have been allowed to cool naturally. It is necessary to practically burn the nose end, and the liberal grinding generally recommended is to remove the outer burnt crust. The writer wishes to confirm Mr. Gledhill's advice that a wet sandstone is the best, and that if an emery wheel be used it should be before the hardening operation. A number of experiments have been made at Clyde Works with oil and hot water as cooling mediums, but it has been found that compressed air at about 60 lb. pressure gives the best all-round results, though hot water may be advantageously used when hard materials are to be cut. It is a good plan for the smith to leave bent tools in his hearth and allow them to soak well through during the night, in order to anneal and take out the forging strains. In the morning the heat-hardening operation should be given to the cutting points only. The smith's hearth is used for the test tools at Clyde Works and has been found the best all-round method. For general work it cannot be beaten, if it is worked by an intelligent man. The hearth must be supplemented by an arrangement for air-cooling connected with the

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blast-pipe. The American gas-furnace, which is useful for cast-steel cutters, will not give the necessary heat for high-speed steel. A gas- and air-furnace for twist drills and smaller milling cutters is very convenient, and must be accompanied by a suitable set of air-cooling tubes. Twist drills should be both heated and cooled vertically to prevent warping. The writer had to harden some Mushet high speed milling cutters 14 in. diameter, also a number of cutters 12 in. long by 5 in. diameter, and the problem was to obtain sufficient heat without damaging the teeth. This would not have been possible in a smith's hearth, and none of the gas-furnaces avilable were large enough. The following method was devised: Some lead was melted in a crucible melting-pot, and the cutter, after being well soaked through in a supplementary furnace, was lowered by means of a bar through the hole of the cutter into the molten lead, and left there until the necessary heat was obtained. In the case of the 14-in. diameter cutter a special pot was made and supported on three brick piers. Another hardening problem was solved as follows: A Mushet high-speed twist drill, $4\frac{7}{8}$ in. diameter by 24 in. long, had to be hardened. A cyclone gas-furnace was used, but after sand-blasting the drill it was found to have two longitudinal cracks up each side. So it was decided to harden the replace drill out of the smith's hearth; it was soaked for twelve hours and finally brought to a full yellow heat, the drill being constantly turned round in the fire. When the desired heat was obtained, it was placed in a specially prepared tube connected with an air-blast.

The testing at Clyde Works of the old Mushet steel was done on a 14-in. lathe, with 4-in. belts driven by a 19-h.p. gas-engine. Mushet high-speed steel soon showed this lathe insufficient in power, so a 16-in. high-speed lathe with five cones, 6 in. wide, was obtained. The line shaft was arranged with two pulleys so that ten speeds are available, and by varying the gas-engine from 160 to 210 revolutions practically all desired speeds can be obtained on the gradually diminishing shafts. The 19-h.p. engine soon had to be replaced with a 36-h.p. engine. All the steel shafts used are of open-hearth steel, made at the writer's works, and cast-iron only of the hardest degrees obtainable is used. At first the softest qualities of steel were operated upon, but such quantities were turned away before

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any conclusions could be drawn that only hard and extremely hard tempers are now used. A specimen test-sheet as used by the writer is reproduced:—

Tool.	Shaft.	Diameter.	Revolutions.	Feet per Minute.	Traverse.	Length.	Cut.	Result.	Remarks.
		In.							
Mushet high-speed, 1 in. sq.....	Hard steel.	13	12	40	7/16	22½	1¼	Tool good.	30 min.
					Re	movin	g 5.	2 lb.	
Mushet high-speed, 1¼ in. sq.....	Hard steel.	12½	15	45	⅛	per mi 18¾	nut 1¼	e. Tool good.	10 min.
					Re	movin	g 1	2.15 lb.	
Mushet high-speed, 1¼ in. sq.....	Oil-hardened axle.	8⅞	24	56	⅛	per mi 13 ft. 10 in.	nut ⅜	e.	Without a grind.

In turning up Mushet high-speed steel blanks for twist drills, Mr. Gledhill's statement can be confirmed about the finish obtainable with high-speed steel if a fine feed be used even at the highest cutting speeds. It is the custom at Clyde Works to turn these blanks at from 80 to 160 ft. per minute. The finishing cuts are done with feeds of 92 cuts per inch of traverse, and this leaves an extremely smooth surface. For the finish on such articles as piston-rods, where a scraping tool is required, the writer strongly recommends a modification of a high-speed steel, a special one being made at Clyde Works, viz.: Regarding twist drills made of high-speed steel, the absolute necessity of the points being ground on a pointing machine ought to be thoroughly realized. A powerful testing machine has been obtained by the writer's firm, it being specially built by Messrs. Tangyes, Ltd. The drill-head is fixed on a slide supported at each end by powerful uprights. The spindle is 3-in. diameter, and is speeded to run at from 78 to 526 rev. per minute. The feeds are from 0.79 to 17.02 in. per minute. It has been found possible to drill at the following speeds:—

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Class of Steel.	Size of Drill.	Nature of Material.	Revolutions per Minute.	Revolutions per Inch of Feed.	Depth of Hole.	Feed in Inches per Minute.
	Inches.				Inches.	
Mushet high-speed.....	$1\frac{5}{8}$	Cast-iron and very hard steel.	181	30.9	3	5.85
Mushet high-speed.....	1	Cast-iron.	269	49.2	3	5.47
Mushet high-speed.....	$\frac{3}{16}$	Very hard steel.	449	49.2	3	9.13
Mushet high-speed.....	$\frac{1}{16}$	Very hard steel.	526	49.2	3	10 69
Mushet high-speed.....	$\frac{1}{16}$	Cast-iron.	526	38.6	3	13 62
Mushet high-speed.....	$\frac{1}{2}$	Cast-iron.	526	30.9	3	17.02*

* At the latter feed of 17.02 in. per minute, a large number of holes have been drilled consecutively.

The practice in making Mushet high-speed steel twist drills is to cut the flutes with the cutters revolving at the rate of 80 ft. per minute.

A cordial vote of thanks having been passed to Mr. Gledhill, Mr. Windsor Richards then announced that the papers by Mr. E. Demenge, and by Mr. Andrew McWilliam and Mr. W. H. Hatfield, Sheffield, and the report on carbon and phosphorus would be taken as read, and proposed votes of thanks to the authors, which were carried unanimously.

Votes of Thanks.

Mr. E. Windsor Richards, Past-President, proposed a vote of thanks to the President of the American Reception Committee, Mr. John Fritz; to the Chairman of the Executive Committee, Mr. Charles Kirchhoff, and the Hon. Secretary, Mr. Theodore Dwight; to the Members of the Reception Committee, and also to the Local Committees in New York, Philadelphia, Washington, Pittsburg, Cleveland, Buffalo, St. Louis and Chicago, for the great cordiality of the welcome extended, and for the arrangements so ably planned and so successfully carried out for the convenience, instruction and pleasure of the members during the meeting.

The vote of thanks was seconded by Mr. Bennett H. Brough, Secretary, and acknowledged by Mr. George W. Maynard, Vice-Chairman of the New York Reception Committee.

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Sir James Kitson, Bart., M.P., Past-President, proposed that the best thanks of the meeting be given to Mr. Carnegie for his able conduct in the chair. Those who preceded him in the chair were conscious of the fact that it was not the simple duty of conducting the proceedings that laid a burden upon the Chairman. He had a good deal to organize, and he had a good deal to arrange. It was due, in the first place, to Mr. Carnegie's initiative that the Institute had the great honor and the great pleasure of being invited to visit the United States. Reference had been made by Mr. Gayley in his paper to the creation of energy. It was perhaps hardly necessary to attempt the creation of energy in the United States amongst any section of her citizens, least of all amongst that section which controls the magnificent iron trade. It was, however, clear that Mr. Carnegie had succeeded in energizing the energetic, and although he claimed that he owed success to the fact that be had found out and used men much cleverer than himself, he had nevertheless been so successful in this organization, and also in his management of the Institute's affairs here, that members would join in voting that the best thanks be given to him for his conduct on this occasion.

The vote of thanks was seconded by Mr. E. P. Martin, Past-President, and carried with acclamation.

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SUBJECT TO REVISION.

[TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.]

Superficial Blackening and Discoloration of Rocks, Especially in Desert Regions.

Postscript to the paper read by Prof. William P. Blake at the Lake Superior meeting, September, 1904.

POSTSCRIPT.*—Since the publication of my paper upon the blackening of the surface of rocks in desert regions, I have noted that similar views of the origin of superficial discoloration in the case of certain rocks from the Salt Lake basin of Utah have been expressed by Prof. George P. Merrill. In the pamphlet[1] he described the brown-coated weathered boulders from Tooele county, reaching the conclusion that the discoloration was due to the solution of the manganese and iron compounds in the interior of the boulder, while they were in the water, and the gradual bringing of this material to the surface through capillarity and its oxidation when exposed.

* Communication to the Secretary received February 28, 1905.

[1] *Bulletin U. S. Geological Survey*, No. 150.

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